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“Porphyry ore body zonality for the mine planning in context of processing performance”

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Master´s thesis which has been submitted for Master of Science degree

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Abstract of master's thesis

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Abstract

Mikheevskoye project is a porphyry Cu-Mo open pit mine located in Chelyabinsk region, Russia. Ore extraction started in 2011 and mineral processing started in late 2013. Mikheevskoye project is owned by the Russian Copper Company.

This study examines the effect of hydrothermal alteration zonality and geometallurgical ore body zonality on the mine planning and plant feed quality forecast. The study was conducted at Russian business unit of Outotec, which operates part of the processing plant in Mikheevskoye project.

The empirical part of the study was conducted in October 2013 - January 2014. Geological data for the study was obtained from Outotec office and Russian Copper Company geologists. Some geological data was collected through sampling campaign in the Mikheevskoye open pit. Additional data was gathered through the questionnaire which investigated how processing engineers working on site view the ore body. A questionnaire was distributed among Outotec and Russian Copper Company process engineers.

The results revealed that mine scheduling based on the geometallurgical zoning is potentially possible and feasible in case of porphyry ore deposits. In this case, twelve geometallurgical zones were determined theoretically. Application of hydrothermal alteration zonality helped improve forecast feed grade quality. Based on the results of this study, it was recommended to conduct additional exploration drilling, evaluate the process performance of the samples retrieved in the drilling and to update the model developed in this study accordingly. One of the key findings of the study was estimation of the new payback time for the project on the basis of current market situation.

Keywords porphyry ore, geometallurgy, mineral processing, copper, Russia, hydrothermal alteration zonning

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Abstract of master's thesis

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Tiivistelmä

Mikheevskoye projekti on Venäjällä, Chelyabinskin alueella sijaitseva porphyry Cu-Mo malmin avolouhos. Malmilouhinta alueella alkoi vuonna 2011 ja rikastamon toiminta vuoden 2013 lopussa. Mikheevskoye malmiesiintymä ja rikastuslaitos ovat Russian Copper Companyn (RCC, Venäjän kupariteollisuus) omaisuutta.

Tässä tutkimuksessa tarkastellaan vesiliukenemisesta ja lämpötilagradientista syntyneiden muuttumisvyöhykkeiden sekä mineraalivyöhykkeiden (geometallurgiset) vaikutusta louhintasuunnitelmaan ja ennustettavaan rikastamon syötön laatuun. Tutkimusta toteutettiin Outotecin Venäjän alueyksikössä; sama Outotecin yksikkö vastaa vaahdotus- ja vedenpoistopiirin operoinnista Mikheevskoye projektissa.

Tutkimuksen kokeellinen osuus toteutettiin lokakuussa 2013 – tammikuussa 2014. Geologinen tieto tuli Outotecin ja RCC:n geologeilta. Osa geologisesta tiedosta oli kerätty paikan päällä avolouhoksesta näytteenottokampanjan merkeissä. Lisätiedot kerättiin kyselyllä, joka tutkii prosessi-insinöörien ymmärrystä malmiesiintymän piirteistä. Kysely toteutettiin Outotecin ja RCC:n prosessi-insinöörien keskuudessa.

Tutkimuksen tulokset osoittivat, että louhinnan suunnittelu perustuen mineraali- tai geometallurgisiin vyöhykkeisiin on mahdollista käytännössä ja on myös taloudellisesti kannattavaa porhyrymalmiesiintymissä. Tässä tapauksessa 12 teoreettista mineraalivyöhykettä oli otettu käyttöön. Vesiliukenemisesta ja lämpötilagradientista syntyneiden muuttumisvyöhykkeiden huomioon ottaminen auttoi tarkentamaan rikastamon syötteen laadun ennustetta. Tutkimuksen tulosten pohjalta suositellaan tuotantokairausten toteuttamista, niistä kerättyjen näytteiden analysointia prosessikäyttäytymisen osalta sekä tässä tutkimuksessa kehitetyn mallin päivittämistä ko. tulosten pohjalta. Tämän tutkimuksen yksi keskeisiä tuloksia oli uusi arvio projektin takaisinmaksuajasta nykyisten metallihintojen pohjalta.

Avainsanat porphyry malmi, geometallurgia, rikastus, kupari, Venäjä, hydrotermiset muuttumisvyöhykkeet

Foreword

I would like to thank the instructors Markku Meriläinen and *Pekka Loven for the wide support and cooperation in such a challenging topic. I would also like to express my gratitude to my supervisor prof. Mikael Rinne for his engagement and useful remarks.*

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Espoo, 30.06.2014

Viktor Lishchuk

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Contents

List of symbols and abbreviations

AA	Atomic absorption
Ag	Silver
Au	Gold
bn	Bornite
CAPEX_M	Capital costs of mine establishment
CAPEX_PP	Capital cost of processing plant construction
cp	Chalcopyrite
Cu	Copper
cv	Pyrite-covellite
E	East
en	Pyrite-enargite
GOK	Mining and beneficiation plant (english) = горнообогатительный комбинат (russian)
Mo	Molibdenium
N	North
NE	North-East
NPV	Net Present Value
NSR	Net Smlter Return
OPEX_M	Operating cost of running the mine
OPEX_PP	Operating cost of the processing plant
PCA	X-ray spectrographic
py	Pyrite
QTZ	Quartz
SSE	South-South-East
VAT	Value Added Tax
XRF	X-ray fluorescence analysis

1. Introduction

Geological model and mineral resources exstimation of the ore body must be done before or during a feasibility study is made. Such a model is created for the resource/reserves estimation. Resource/reserves estimation provides data for the feasibility study.

The block model has to be created for the feasibility evaluation. The block model comprises of blocks where each block has indirectly assigned costs (cost of extraction and further processing) and revenue (price of contained metals and useful materials). Therefore, location of each block and block's value (costs and revenues) are the main parameters in decision making for further mine planning and feasibility study.

Ore body boundaries outline the limits of the block model and are mainly defined on the basis of grade. Such approach is appropriate when grades are high and difference between revenue and expenses is much higher than with lower grade ores. With lower grade ores profits are achieved mainly due to huge production scale. Therefore, ore body boundaries can not rely only on ore grades. Metallurgical data has to be incorporated into geological model (block model) in order to bring more certainty into block costs and revenues where it is possible. Geometallurgical zoning is the name for the above described approach which leads to the spatial predictive model for mineral processing plants (Lamberg, 2011).

Costs

Each block's cost (Figure 1) is defined by capital costs of mine establishment (CAPEX_M), capital cost of processing plant construction (CAPEX_PP), operating cost of running the mine (OPEX_M) and operating cost of the processing plant (OPEX_PP). Traditional approach assumes that CAPEX_M and CAPEX_PP are constant values for each mineable block, OPEX_M is variable value for each block depending on its location and OPEX_PP is a constant value. However, due to low (and constantly decreasing) grades of mineral deposits worldwide, OPEX_PP should be treated as a variable value.

Geometallurgical parameters (such as hardness, content of further treatment penalty materials, other requirements) have strong impact on the processing cost of the ore and thus on the final cost of the concentrate production. Modern geological modeling tools allow us to take into account these parameters and specify a number of metallurgical ore types. This leads to the more precise cost model within the block model framework. Therefore, OPEX_PP should be treated as a variable (Figure 1). Increased precision of the cost estimation of the block allows more accurate mine planning and plant feed forecasting (this results in reduced operating costs).

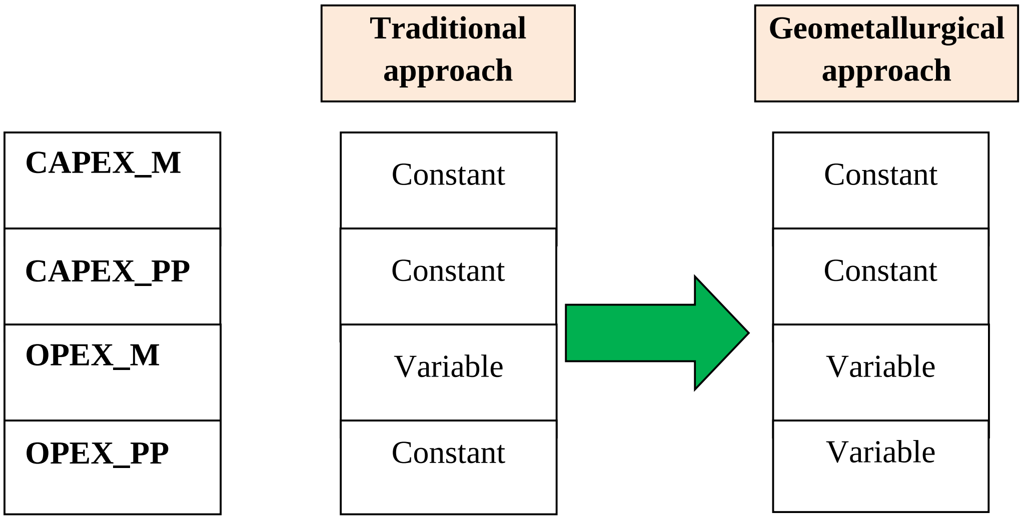

Figure 1 Geometallurgical approach.

Revenues

Porphyry Cu deposits can be studied through hydrothermal alteration zoning, where economic sulphide zones are associated with certain alteration zones. Although, information on hydrothermal alterations is not crucial for the feasibility study, it may play a significant role in mine planning and extraction scheduling. Hydrothermal alteration zonality was used in this study to create more precise metal distribution model and hence help to define more precisely potential revenues from the extracted blocks. Paying enough attention to the hydrothermal alteration zonal pattern **(**Salehian and Ghaderi, 2010**)** and thus mineralization may reduce exploration costs due to improved planning of the geological exploration campaign.

Two main issues can be considered from the perspective of processing plant feed quality forecast in this study: applying hydrothermal alteration zonality over ore body for better metal distribution modeling; and applying geometallurgical zonality over ore body for the improved OPEX_PP cost estimate.

Feed quality

Three main parameters of the feed quality could be used for the geometallurgical zoning in case of the Mikheevskoye Cu-porphyry deposit: oxides content, hardness, and iron presence.

One of the current disputes between mining and concentrator departments lays in lack of clearness in the separation of border line between oxidized and transitional ore zones. Meanwhile oxidized ore is completely prohibited to be fed into the process due to its inhibiting effect on the recovery of the primary sulphide ore. Transitional ore has no such limitations and is seen by the mining department as an ore type suitable for the input into the process and shuffeling due to its increased chemical Cu content and lower hardness and density. The last two features help to keep the tonnage feed into the process high with less explosives consumption.

Wearness of the equipment and particle size distribution of the pulp are heavily dependent on hardness of the fed ore. Accurate hardness zonality would allow to predict energy consumption of the crusher and mills; wearness of the pumps and linings; and reagent consumption due to surface area of the hard particles in the pulp.

Presence of the hematite and magnetite in the feed requires additional processing for the feed. First, magnetic separation is applied to separate iron from the fed ore. Secondly, additional processing of the pulp is required for the iron suppression.

1.1. Aims

The aim of the Master thesis is to improve performance of the processing plant through the better understanding of the mine planning of the Cu-porphyry deposits. The aim of this study can be reached by applying hydrothermal alteration zoning in geological block modeling and geometallurgical zonality in operational cost estimate.

Improved understanding of the mine planning is crusial for creating a forecasting tool for the mineral processing performance. This particular study is concerned with Mikheevsky ore deposit. The final product (forecasting tool) of this study should be an “improved mining schedule”, which is a constructive integration of geology, mining and processing aspects based on principles of porphyry copper ore zonality geometallurgical zonality. This model is expected to pay significant financial dividends through maximizing output and efficiency of the processing plant in both short and long-term (Alruiz et al, 2009) and be beneficial for mine planning.

Better performance of the Mikheevskoye production chain under improved schedule will suggest opportuinities for applying proposed method to other Cu porphyry projects worldwide.

1.2. Objectives and fieldwork

Objectives for this thesis were derived from the geometallurgical program proposed in (Lamberg, 2011).

- Investigate feed quality needs of concentrator process (separate needs for comminution (Metso) and flotation (Outotec) departments).
- Collect up-to-date geological information about the deposit;
- Conduct sampling campaign;
- Collect up-to-date topographic data from surveying;
- Model zonality of the ore body based on ore performance in the process;
- Run open pit optimization;
- Develop optional open pit design based on ore zonality;
- Develop mining plan and extraction schedule;
- Estimate cost efficiency of the proposed solution.

Mikheevskoye deposit was used as a testing area for this thesis. All the geomodelling was performed in Surpac 6.3 software, including modeling and analysis of the: geological database, solids, block model, variograms, values estimation (interpolation), open pit modeling and open pit design.

1.3. Porphyry deposits

1.3.1. Definition

Porphyry copper deposits are large (greater than 100 Mt), low- to moderate-grade (0,3–2,0 % copper) disseminated, breccia and vein-hosted copper deposits hosted in altered and genetically-related granitoid porphyry intrusions and adjacent wall rocks, and include associated weathered products. Porphyry copper deposits are associated with shallowly emplaced (less than 10 km) stocks and dikes and underlying plutons and batholiths and commonly show locally broadly coeval volcanism.

The most common host rocks are quartz monazite and granodiorite. Copper is the dominant metal in most of porphyry copper deposits. In many deposits, Mo and Ag can be important by-products.

According to (Silitoe, 2010), porphyry Cu systems nowadays are responsible for the nearly three-quarters of the world's Cu (including the world's largest known exploitable concentrations of Cu), half the Mo, perhaps one-fifth of the Au, most of the Re, and minor amounts of other metals (Ag, Pd, Te, Se, Bi, Zn, and Pb).

Porphyry Cu deposits have significant impact not only on the economics but also on the social life, due to their long mine lives and high production rates (John et al, 2010). They are of great economic significance in research of porphyry copper deposits throughout the world (Xia Bin et al, 2003). Porphyry deposits occur throughout the world in a series of extensive, relatively narrow, linear metallogenic provinces (Sinclair, 2007).

1.3.2. Supply

Porphyry Cu systems are nowadays the primary source of the world's Cu and remain one of the main targets for the global mineral exploration industry **(**Holliday and Cooke , 2007**)**.

Most porphyry Cu deposits contain minor economic quantities of Mo and Au. The huge tonnage of mined ores of porphyry Cu deposits provides large quantities of Mo and Au as by-products. **(**Shafiei and Shahabpour, 2012**)** Porphyry Cu deposits are the largest source of Cu – 60% and resource of Cu – 65% (John et al, 2010; Xia Bin et al, 2003; Sinclair, 2007**)**. Between half and 95 % of the world's Mo (Sinclair, 2007), one fifth of the Au (e.g., according to **(**Volkov et al, 2006**),** porphyry deposits provide approximately 20% of the Au in the United States) and most of the Re is also supplied by the porphyry Cu systems. Some other elements can be a by-product of the Cu deposits' exploitation: Ag, Pd, Te, Se, Bi, Zn, W, In, Pt, Pd, Se, and Pb (Silitoe, 2010; Sinclair, 2007).
The dominant Cu minerals in hypogene ore are chalcopyrite and bornite. Bornite occurs in 75% of deposits. Molibdenite, a molybdenum mineral, occurs in 70% of deposits. Molibdenite is the only molybdenium mineral. Au occurs in 30% of deposits (John et al, 2010).

More details on the Cu(Au,Mo)-porphyry deposits could be found in the (Lefevre and Harold, 1987).

1.3.3. Origin

Porphyry deposits range in age from Archean to Recent, although most economic deposits are Jurassic or younger (Sinclair, 2007).

Porphyry Cu systems are initiated by injection of oxidized magma saturated with S- and metal-rich, aqueous fluids from cupolas on the tops of the subjacent parental plutons (Silitoe, 2010). Thus, most of the Cu minerals are deposited along thin fractures, zones of brecciation, and within larger veins, called stockwork veins (McLemore, 2008). Best Cu porphyry ore bodies are presented by the early formed features; however, late-stage alteration overprints may remove Cu and Au. This pattern can be illustrated as following (Figure 2) (Silitoe, 2010).

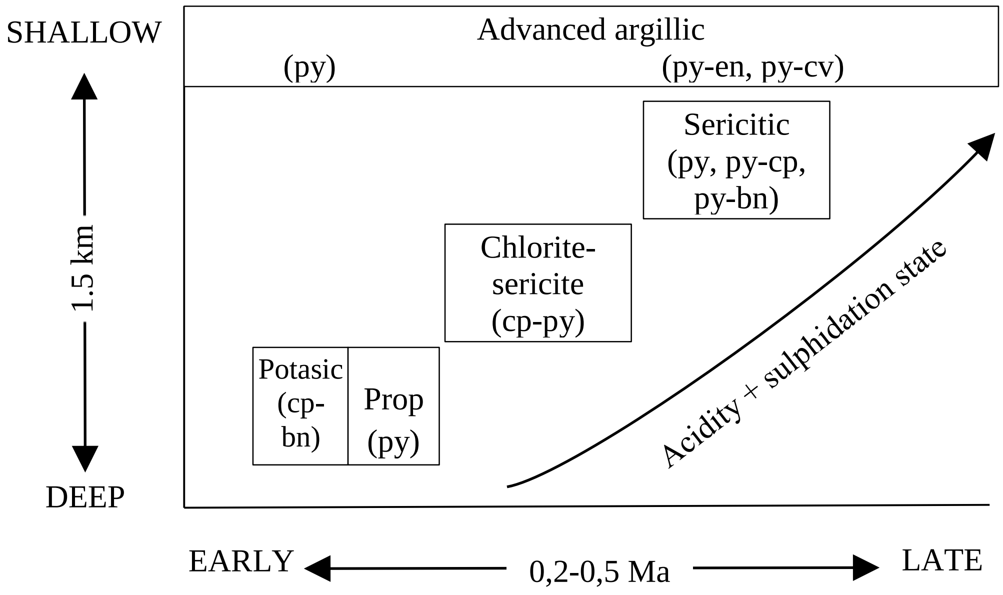

Figure 2 Generalized alteration-mineralization sequence in porphyry Cu systems[1](Silitoe, 2010).

[1] Sericitic alteration is also often called phyllic; and chalcopyrite (cp), bornite (bn), pyrite-enargite (en) or pyrite-covellite (cv), pyrite (py).

1.3.4. Grades

Cu grades in Cu porphyry deposits are quite low and vary from 0,1% to over 1,5% with 2-5% pyrite, Mo – 0,01-0,04% (sometimes up to 0.3% (Sinclair, 2007)) and Au – 0,0-2,0 g/t (Silitoe, 2010; Sinclair, 2007). Relatively low mineralization can sometimes be enhanced through supergene enrichment. Supergene enrichment is a weathering process and occurs when rocks with high pyrite content come into contact with water in an oxidizing environment (McLemore, 2008)**.**

1.3.5. Shapes

Circular and elliptical shapes in plan view are typical shapes for the undeformed deposits. In cross section, ore zones vary from cylindrical shells with altered, but low-grade, interiors referred to as "barren" cores, to inverted cups around barren cores, to multiple domes or inverted cups, and to vertically elongate, elliptical shapes (John et al, 2010). Statistics of areas of ore, sulphides and altered rock in porphyry Cu deposits is following:

Table 1 Summary statistics of areas of ore, sulphides and altered rock in porphyry Cu deposits (John et al, 2010)

Statistics	Area of ore	Area of sulphides	Area of alterations
Mean, km^2	1,25	7,4	8,9
Median, km^2	0,6	3,7	5,1
Max, km^2	28	89	82
Min, km^2	0,02	0,18	0,24
Number of deposits	174	173	184

1.3.6. Alteration-mineralization zoning in porphyry Cu deposits

Porphyry Cu deposits quite often demonstrate an alteration-mineralization zoning patern. (Silitoe, 2010; Salehian and Ghaderi, 2010**)**. Zoning patern is usually arranged in a shape of a shell (Silitoe, 1973). For many Cu porphyry deposits it could be stated that alteration zones on a deposit scale consist of an inner potassic zone characterized by K-feldspar and/or biotite (± amphibole ± magnetite ± anhydrite) and an outer zone of propylitic alteration that consists of quartz, chlorite, epidote, calcite and, locally, albite associated with pyrite (Figure 3). Zones of phyllic alteration (quartz + sericite + pyrite) and argillic alteration (quartz + illite + pyrite ± kaolinite ± smectite ± montmorillonite ± calcite) may be part of the zonal pattern between the potassic and propylitic zones, or can be irregular or tabular, younger zones superimposed on older alteration and sulphide assemblages (Bosc and Barrie, 2013).

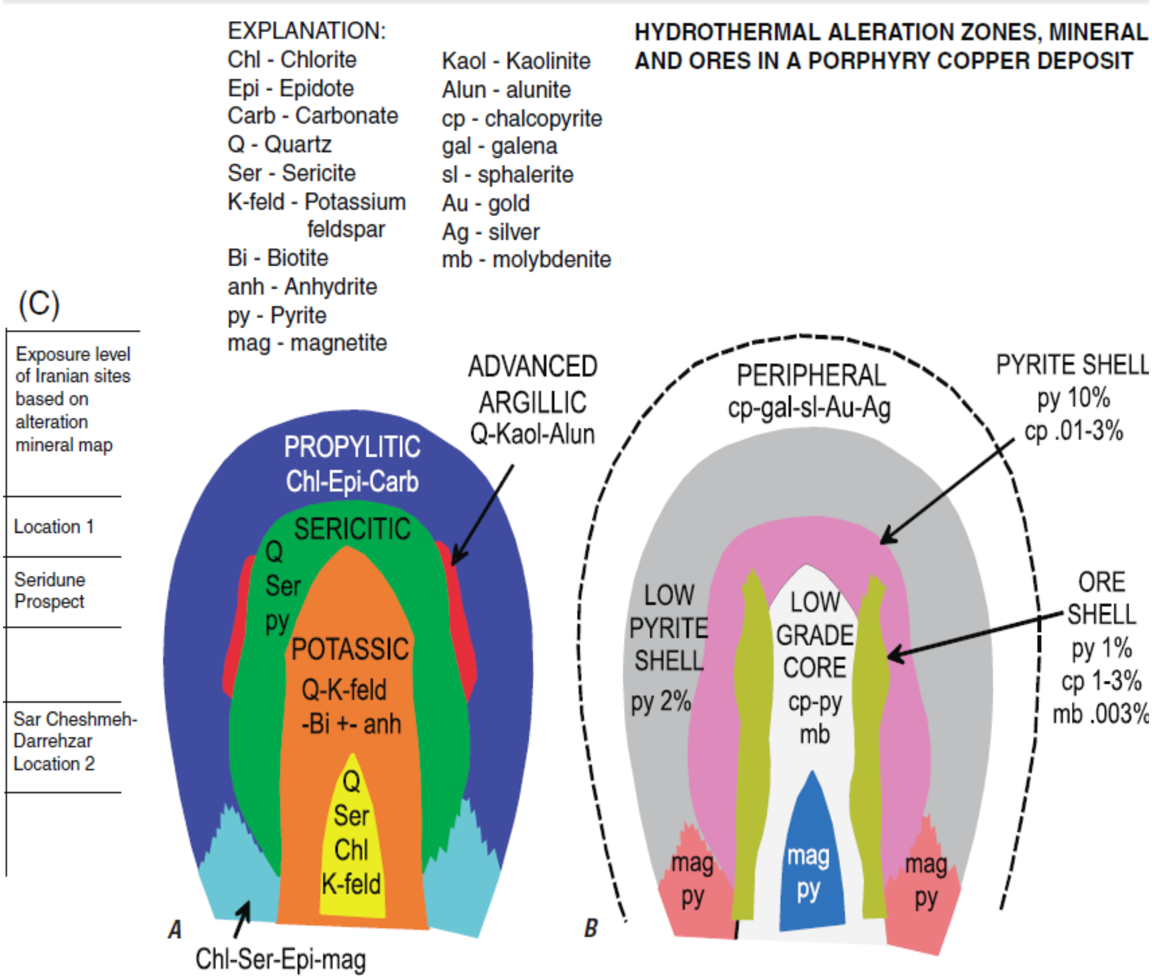

Figure 3 Schematic model of alteration and mineralization of a porphyry copper deposit. (Lowell and Guilbert, 1970)

The highest Cu concentrations are observed within Cu porphyry deposits which maintained their early porphyry phases and potassic alteration assemblages in unmodified shape. This is also relevant for the Au, which tends to be removed and dissipated during the formation of lower temperature, pyrite-bearing alteration assemblages (Silitoe, 2010). It is common that potassic zone is a central zone of the orebody and is represented with the biotite-orthoclase-chalcopyrite mineral assemblage. However, the concentric zoning may be destroyed and replaced by younger quartz-sericite alteration in some deposits (McLemore, 2008). Types, original and replaced minerals of hydrothermal alteration are listed in Table 2.

Table 2 Hydrothermal alteration characteristics typical of porphyry Cu. (Lefevre and Harold,1987)

Alteration	Original mineral	Replaced by	Appearance
Potasic (K-silicate)	Plagioclase	K-feldspare	Rocks look fresh but may have pinkish K-feldspar veinlets and black biotite veinlets and cluster of fine biotite after mafic phenocrysts.
	Hornblende	Fine-graned biotite + rutile + pyrite or sagnetite. Anhydrite	
Sodic-calcic (albitic)	K-feldspar	Oligiclase or albite	Rocks are hard and dull white. Biotite is absent. Veinlets of actinolite, epidote and hematite have hard, white alteration holes.
	Biotite	Actinolite + aphene	
Phyllic (quartz-sericite)	Plagioclase	Sericite	Rocks are soft and dull to lustrous white. Pyrite veinlets have distinct, soft translucent gray, sericite haloes. Tourmaline rosettes may be present
	Hornblende and biotite	Sericite + chlorite + rutile + pyrite	
Propylitic	Plagioclase	Albite or oligoclase + epidote or calcite	Rocks are hard and dull greenish gray. Veinelts of pyrite or chlorite and epidote lack prominent alteration haloes.
	Hornblende and biotite	Chlorite + rutile + magnetite or pyrite	
Argillic	Plagioclase	Clay + sericite	Rocks are soft and white. Tongue will stick to clay altered minerals
	Mafic minerals	Clay + sericite + chlorite + pyrite	
High alumina (alsic, advanced argillic)	All original and earlier hydrothermal minerals converted to pyrophylite, alunite, andalusite, corundum and diaspore with veriable amounts of clay and sericite		Rocks are light colored and moderate soft.

1.4. Description of the Mikheevsky project

The geometallurgical approach in this study has been applied over the case of Mikheevskoye project operated by Russian Copper Company in cooperation with two Finnish companies (Metso and Outotec). Mikheevskoye project operates a porphyry Cu-Mo ore body.

Mikheevskoye deposit is 112nd in the world by Cu content (more details could be found in (Scott et al, 2011; Sinclair, 2007), and 94th by Cu grade. There are 70 (out of 150 the biggest) world's copper deposits which contain Au and Mikheevskoye is 50th by Au content (Scott et al, 2011).

Mikheevskoye project started its mining activities in 2011. It accounts for 1,70% Russian Cu resources (with a Cu grade of 0,44%). Mikheevskoye deposit is considered to be the largest (Mo, Au)-Cu porphyry deposit in the Ural **(**Grabezhev, 2012). It also accounts for the highest Re content in Ural **(**Grabezhev, 2013**).** Mikheevskoye project is owned by Russian Copper Company which is the third biggest Cu producer in Russia. RCC produces 188 kt/a of Cu. (the largest Cu producer in Russia is OOO "UGMK" with 379 kt/a and second largest is OAO "Norilsk Nickel" with 357 kt/a) (IAC Mineral /ИФЦ Минерал, 2011).

1.4.1 Property description, location and accessibility

Mikheevskoye deposit is located in Chelyabinsk region, Russian Federation on the territory of the Varna municipality on the border with Kartaly municipality.

Mikheevskoye got its name in 1983 from the abandoned farm located nearby (Bulatov et al, 2010). Local resident, A.D. Shybanov has found pieces of the oxidized copper-magnetite ore in 1952 near the Novonikolayevka village, located on the northern side of the Karataly-Ajat river. This was the beginning of the active geological exploration of this region. (Novikov et al, 2010). The industrial importance of the deposit was indicated in 1986-1987.

Since 2007 100% of the "Mikheevskoye GOK" stocks belong to "Russian Copper Company" (RCC) - Yekaterinburg, Russia (Bulatov et al, 2010; IMC Economic and Energy Consulting Limited, 2008). License for natural resources "ЧЕЛ 12003 ТЭ" is valid until 30 August 2022 and was issued for the "Mikheevsky GOK" on 11 December 2003. This license allows geological exploration and excavation of the copper-porphyry ores from the open pit. License covers area of approximately 1,86 km^2 (Alferov et al, 2010).

1.4.2. Exploration

The exploration network created by 2011 had density of 70 × 90 m (sometimes up to 50 × 50 m); distance between drill holes was 20 - 135 m, and distance between lines of drill holes was 45 - 170 m. Resolution of drill holes network was decreasing (to the 100 × 100 m) with the depth and in the southern zone **(**Scott et al, 2011).

Research has revealed that deposit dimensions were 3000 × (200–750) m (Alferov et al, 2010). The deepest drill hole was M011 with depth of 500.1 m and had passed primary ore in the depth range of 13,9-489,0 m (Bulatov et al, 2010). GPS measurements have shown that all the actual drill holes were located within 1 m from the theoretical drill holes' location **(**Scott et al, 2011).

According to **(**Scott et al, 2011), data from the 267 drill holes bored before 2000, where 164 were diamond drilling, and the rest were rotary drilling, had been lost. Therefore, total of approximately 470 drill holes (including lost data) were explored during 1952 – 2008.

Mineral resources at Mikheevskoye deposit were proved to be C_1 and C_2 category (Table 3), (Novikov et al, 2010; Shargorodskiy et al, 2007).

Table 3 Comparison of the Russian and JORC systems. Based on (Henley, 2004)

JORC	Proved reserves / Measured resources	Probable reserves / Indicated resources	Inferred resources	Unclassified
Russian system	**A+B**			
	C_1	**C_1**		
		C_2	**C_2**	
			P_1	**P_1**
				P_2+P_3

It is possible that, the sulphide zone in the North body closes off both laterally and in depth. This means that Mikheevskoye system can have undergone profound tilting and is likely to be bowl-shaped. Drilling of the northern part has shown that base of the mineralization is dipping to the south. It seems most likely that the entire system has been tilted northwards and, hence, plunges gently southwards. Therefore, it is possible that there might be a chalcopyrite-bornite core under Central and Southern bodies. Alternatively, transverse faulting may have raised or depressed these bodies relative to the North body and to one another, resulting in the possibility that a bornite-bearing core has been eroded to leave only the basal part of the chalcopyrite zone. **(**Scott et al, 2011).

1.4.3. Geological setting

It was estimated on the basis of U-Pb SHRIMP-II zircon ages of granitoids that Mikheevskoye Cu-porphyry deposits is 356 ± 6 Myears (Grabezhev, 2012). The Mikheevskoye porphyry prospect is hosted by Late Devonian volcano-sedimentary rocks, of andesitic to basaltic composition and overlying Early Carboniferous basaltic volcanic rocks (Grabezhev, 2007). The Paleozoic host rocks are cut by diorite and quartz diorite intrusions of assumed Early Carboniferous age. A thin cover of Quaternary alluvium overlies the prospect area.

The Mikheevskoye prospect is centered on several closely spaced bodies of quartz diorite and diorite porphyry, which are dyke-like in shape. The bodies strike north-northeast over a distance of approximately 3 km, and dip steeply eastwards. The dykes appear to merge southwards with a larger intrusion of reportedly similar composition. It has been proposed that a fault zone, possibly a strike-slip structure, localizes the intrusive belt. Transverse post-mineralization faults may cut and offset the prospect area, giving rise to separation of the <u>North, Central and South</u> parts of the deposit.

Alterations and mineralization encompass the zone of dyke-like intrusions, and give rise to broadly coincident top-of-bedrock copper geochemical and induced polarization chargeability anomalies. A thin zone of supergene oxidation contributes to the anomalous geochemical copper values. The geochemical results also define a series of zinc-lead-copper-arsenic-silver-gold concentrations along the sides of the copper-gold prospect **(**Scott et al, 2011).

1.4.4. Alterations

Metasomatic halos of the Mikheevskoye deposit correspond to the morphology of the dykes, faults and fractures. The total length of the halo of the Mikheevskoye deposit is

6 km and its width is 0,8 km. There is a certain sequence in a frequent alternation of the rocks of a different composition (from inner zones to the outside zones): sericitized rocks - sericitized and chloritized rock - propylitized rocks (Novikov et al, 2010).

Sericitized rocks are represented by two paragenetic generations of alterations: the first one (sericite, quartz) is spread in the central part of the Mikheevskoye deposit, the second one (sericite, quartz, carbonate) is common in the northern and southern flanks and forms a linear halo. Relict minerals of sericitized rocks are chlorite and de-anorthitized plagioclase (up to albite-oligoclase).

Observations have shown that more acid altered rocks have stronger sericitic alteration. In diorites and plagiogranites, mafic phenocrysts are usually completely cericitized, although relicts of chlorite and amphibole still can be found. Mafic phenocrysts are completely sericitized in plagiogranites-porphyries, and plagioclase phenocrysts are sericitized partly; plagioclase of the ground mass is not replaced. Phenocryst of the basic plagioclase is replaced in the first order in the basic effusives and bulk composition in a sericitization zone. Sericite is developing with strong alterations in femic minerals of the bulk rocks. In phenocrysts of volcanic rocks, sericite is developing with preserving crystals zone structure and replaces saussurite, epidote, chlorite (Novikov et al, 2010).

Carbonate-sericite-paragonite-quartz metasomatic rocks are developing mainly by granitoids - diorite porphyry, granodiorite porphyry, rarely by ultramafic rocks. Their thickness is up to 100 - 130 m. Metasomatites are composed of quartz (25-50%), calcium carbonate (3-8%), pyrite (0-5%), variable amounts of sericite and paragonite (up to 10%). Paragonite and sericite are, in some places, in association with albite and kaolinite, calcite (argillic zone) (Novikov et al, 2010).

Propylitized rock formations are represented by actinolite-epidote, epidote-chlorite and chlorite subfacies. They also contain calcium, magnetite, pyrite, and sericite (in subordinate quantity). The presence of low-temperature titanium magnetite (2-3 %) is a characteristic of near-ore propylites. This allows to distinguishing them from greenstone altered rocks, which are similar by mineral parageneses, and map out propylite fields with help of magnetic survey methods. Propylites are the most common metasomatic rocks, which change into locally metamorphosed rocks on the periphery of the halo (Novikov et al, 2010).

Metasomatic chlorite subfacies occupy an intermediate position between the chlorite-epidote propylitic and sericitized rocks in the generalized scheme of zoning metasomatic aureole and reveal gradual transition to the sericitized rocks. Besides chlorite, which prevails in this zone, sericite, calcium carbonate, and sulphides are present in variable amounts. In this zone volcanic rocks and their ground mass are altered by chlorite phenocrysts, and chlorite forms porphyroblastic clusters of 0,5-2,5 mm. It is a colorless or pale green pennine, rarely light brown clinochlore. Chlorite shows a very high iron content of 0,45, whereas propylitic chlorites show only 0,20-0,35.

Argillic zone was outlined and studied within Mikheevskoye deposit. Argillic zone usually means clay rocks of the metasomatic genesis associated with highly sulphidic copper-porphyry and epithermal systems, which include kaolinite (and its polymorph dickite) and montmorillonite. Significantly micaceous rocks (filicides) can be counted as argillic as well as mica in its hydrated shape – illite. The nature of the rock-forming minerals, argillites, indicates a relatively low temperature of formation comparing with the ground mass of metasomatic rocks of porphyry copper deposits.

The boundaries between different types of metasomatic rocks in the Mikheevskoye copper-porphyry deposit are fuzzy and gradual. This is typical for the boundaries between the propylitic, phyllisite (essentially metasomatic sericite) and argillites.

These metasomatic rocks are close to phyllisites and occupy an intermediate position between them and propylites. They are different from propylites due to the absence of carbonates, and are different from the classic phyllisites due to predominance of the chlorite over mica components and a minor presence of a disordered kaolinite.

In general, Mikheevskoye deposit can be characterized as follows (Novikov et al, 2010):

- All the rocks of the Mikheevskoye deposit are characterized by hydrothermal-metasomatic alterations.

- Mikheevskoye deposit is characterized by a relatively low degree of alteration in the bedrock under the influence of metasomatic processes. Intensely substituted rocks (mica-quartz metasomatic) have limited distribution against the general background of low- and mean-alterated rocks due to intense tectonic setting of mineralization zones formation.
- Metasomatic facies are alternating and zones power fluctuates from the first meters down to 150-200 m.
- It is assumed that alterations of the Mikheevskoye deposit occurred as a result of pulsating magmatic and fluid activity of the deep source and relate mainly to the granitoid intrusions of the Mikheevskoye deposit.

1.4.5. Mineralization

Mineralization in the Mikheevskoye deposit is Early Carboniferous in age and it is related to the swarm of the intrusive porphyry dykes within volcano-sedimentary rocks, of andesitic to basaltic composition, and overlying basaltic volcanic rocks. Copper mineralization occurs as chalcopyrite and bornite (Figure 4) as disseminations within the host lithology **(**Scott et al, 2011).

Ore zones of the Mikheevskoye deposit have locally outlined, sometimes not well defined vertical mineral zonality (ore stratification) from the top to the bottom (IMC Economic and Energy Consulting Limited, 2008; Alferov et al, 2010):

1. The top layer colored in red is a layer of the shallow Cainozoic rocks (soil) .
2. Laterite zone (also known as supergene or oxidized zone - oxidized ore) - yellow in Figure 5, Figure 11.
3. Intermediate (oxidized/ cemented) zone - transitional (mouldy) ore, green in Figure 5, Figure 11.
4. Hypogene (fresh) zone - sulphide (rocky) ore, blue in Figure 5, Figure 11.

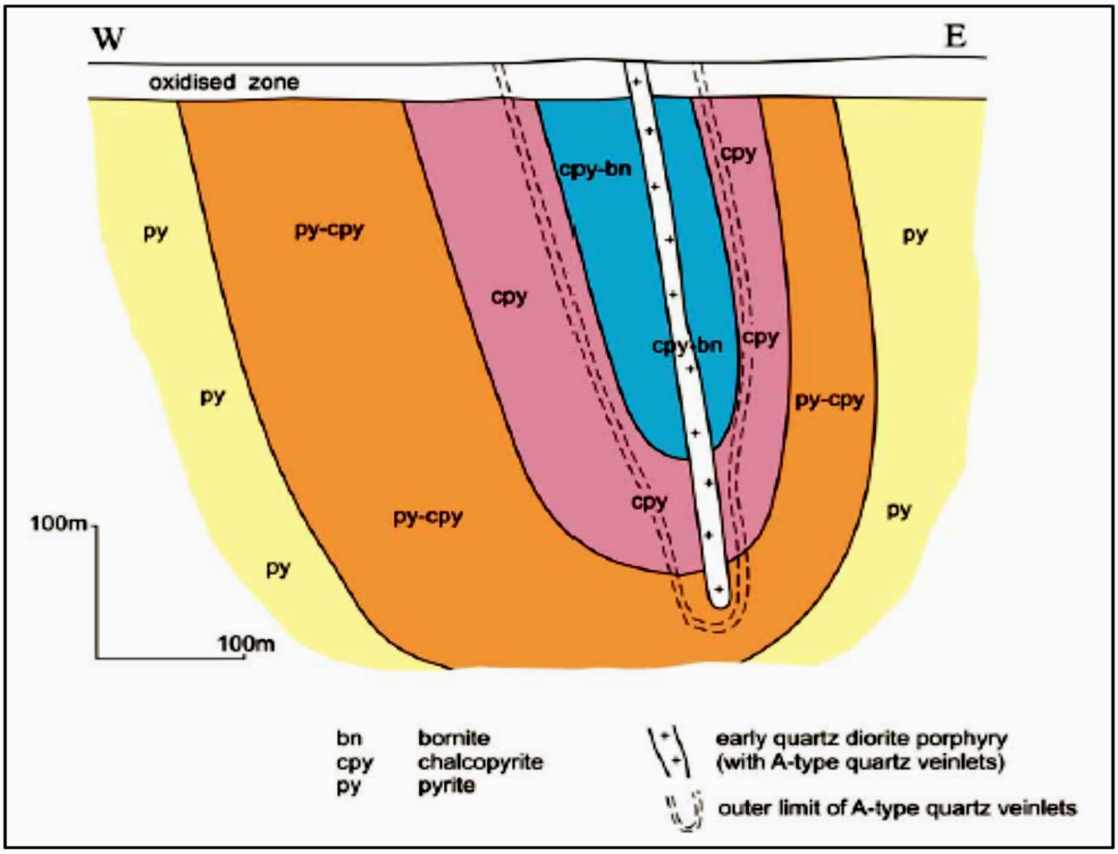

Figure 4 Scheme of the sulphide zonality at Mikheevskoye deposit. (Deter et al, 2006)

Ore from diferent zones can be separated on the basis of several criteria (Table 4).

Table 4 Criteria for the ore zone separation.

	Laterite (Oxidised)	Transitional	Primary
Aggregative state	Clayey	Debris-clayey	Rocky
Visual appearance	Warm colours, pinkish, yellowish, brownish	Cool colors. grayish, greenish, bluish	

Each zone has its specific mineralogical composition (Table 5).

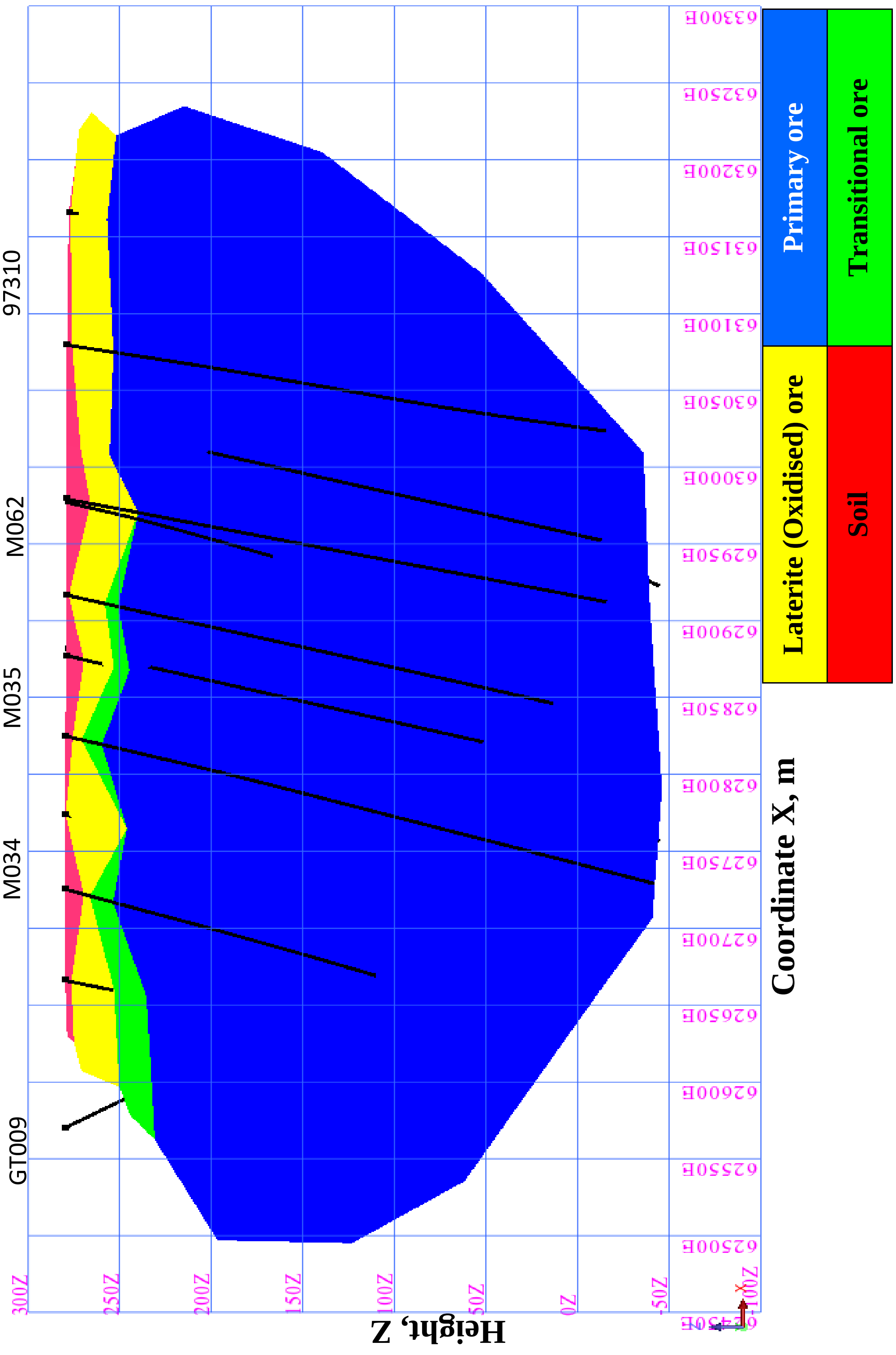

Figure 5 Weathered layers in orebody.

Table 5 Mineralogical composition of the ore at Mikheevskoye deposit. (Alferov et al, 2010; Shargorodskiy et al, 2007)

<table>
<tr><th colspan="2" rowspan="2">Ore type</th><th colspan="2">Type of ore minerals</th></tr>
<tr><th>Basic</th><th>Associated</th></tr>
<tr><td colspan="2">I. Laterite (Oxidised)</td><td>limonite, goethite, malachite</td><td>azurite, covellite</td></tr>
<tr><td colspan="2">II. Transitional</td><td>pyrite, chalcopyrite</td><td>chalcocite, melnicovite, magnetite</td></tr>
<tr><td rowspan="3">III. Sulphide</td><td>Phylic</td><td>bornite, chalcopyrite, magnetite</td><td rowspan="3">sphalerite, galena, molybdenite, pyrrhotite, sphene, ilmenite</td></tr>
<tr><td>Sericitic</td><td>chalcopyrite, magnetite</td></tr>
<tr><td>Propylitic</td><td>chalcopyrite, pyrite</td></tr>
</table>

Cores with high Cu or Au content can be found within three main blocks of mineralization. These cores correspond to the chalcopyrite and bornite or only to chalcopyrite zones. Mineralization is changing into pyrite (locally polymetallic lodes where zinc - lead - copper - arsenic - silver - gold may occur). These changes take place from the center towards the edges of the deposit (IMC Economic and Energy Consulting Limited, 2008).

Block with high Cu content (the average Cu content is 0,8-1,0%) was found in the Northern part of the Mikheevskoye deposit. This was the reason why production started in the Northern part of the deposit.

The general property of the mining stock work has sharp sub-vertical eastern border and fuzzy, rough northern, western and southern borders. Dip of the stock work is southern, eastern and northern respectfully. Therefore, stock work has shape of the deep and elongated cup, which ensures a low level of overburden.

Eastern part of the stock work is the richest. It might be due to clear temperature and geochemical border of the maximum temperature gradients. Deposit has clear lateral geochemical and mineralogical zonality. High Cu, Mo, Au concentrations, within the ore stock work, are edged by pocket and veinlets mineralization.

Soil

A thin cover (0,5 - 5 m) of quaternary alluvium (loam, clay and sand) which overlies the prospect area is called here a soil. Soil layer has to be removed in order to enable further ore excavation. After mining is over, soil can be used for rehabilitation of the demaged areas.

Laterite (Oxidized) zone

There is a lot of confusion an ambiguous terminology used in technical documentation related to the weathered layers of the orebody. Although oxidized ore and laterite is very similar, it is still important to desinguish them: laterite has >70% of clay and oxides have <70% of clay. In this case it is using term laterite would be more accurate. Predominant Cu-minerals in oxidation zone are malachite, chrysocolla and covellite. Enclosing matrix composition comprises quartz, kaolinite, illite, albite, amphibole, chlorite, smectite phases in different proportions; besides that, there are calcite, sericite and talc. Oxidized zone of the deposit has significant quartz content of the upper layers and clay component which increases with the depth (Novikov et al, 2010).

Oxidized ore inherits textural characteristics of the primary sulphide ore and is characterized as sticky, finely disseminated and spotty.

Oxidized zone has an aerial lenticular shape with individual pockets. The greatest thickness of the oxidized zone is in the central part of the deposit (lines 100a, 100 – (Appendix 2, Appendix 3) and the widest part is in the southern block (lines 87, 86 – (Appendix 2, Appendix 3). Thickness of the oxidized ore zone is within range of 1,5 – 83,6 m, with an average of 14,7 m and length of about 3,3 km. The maximum thickness of the oxidized zone is in places where it is developed by propylitized, argillized volcanic and volcanoclastic rocks, the minimum - on granitoids and quartz metasomatites.

Cu distribution in the oxidized ore is irregular, which causes low quality of the extracted material. Distribution of the associated metals is also irregular. (Bulatov et al, 2010)

Differentiation between oxidized and transitional zones is not always clear. However, there is a difference in sulphur content between these two zones. Oxidized zone has very low sulphur content, which is usually <0,1%. Sulphur content increase reflects intersection with the pyrite mineralization in the upper layer of the transitional zone **(**Scott et al, 2011).

Transitional zone

Transitional zone includes friable (soft, loose) sulphide zone and mixed sulphide zone according to Russian nomenclature. The major part of this mineralization is located in the western flank of the deposit where its thickness is about 10 m and more. Thickness is significantly lower in the eastern direction. Length of the ore body is around 2,8 km (Bulatov et al, 2010). Thickness of the lenses is 1,6-45,5 m (14 m on average) (Novikov et al, 2010).

Transitional zone is smaller than oxidized by both weight and volume. The ultimate border between them can be defined on the basis of sulphur content. Oxidized zone has sulphur content < 0,5% according to (Bulatov et al, 2010) or < 0,1% according to **(**Scott et al, 2011).

Transitional ore is missing in some parts of the northern and central blocks of the deposit, which could be a result of initial presence of quartz-containing minerals (siliceous quartz-sericite, quartz sandstone) (Bulatov et al, 2010).

The lower border of the transitional ore is dropping when contacting to the metasomatic alterations of the volcanic rocks and is increasing in zones of silication and in the parts with intensive development of dyke assemblage.

In contrast to the predominantly clay oxidation zone, transitional zone is lithologically represented by clay-detritus gray, greenish-gray, metasomatic alterated rocks with a pyrite impregnation, or silicate uniform rock of the whitish-gray color with white kaolinite veins of the 3-5 mm thickness (Novikov et al, 2010). Kaolinite content in the transitional zone is much lower than in oxidized zone **(**Scott et al, 2011).

Transitional zone has following mineralogical composition: chalcopyrite, pyrite, Ti-magnetite, sphalerite, molybdenite, azurite, malachite, and covellite. The most spread Cu containing mineral is chalcopyrite and much less – covellite. Other metals are represented as follows: Au – 0,002-0,990 g/t, Ag – 0,08-4,57 g/t, Mo – 0,00005-0,044%. Non-metallic minerals are quartz, chlorite, hydrated sericite and kaolin. Sulphide minerals in this zone include pyrite and chalcopyrite (Bulatov et al, 2010).

A secondary enriched zone occurs locally throughout the transitional zone. It is represented by secondary sulphide minerals chalcocite and covellite so that copper grades can sometimes be very high Typical characteristic of transitional ore is an abrupt jump of Ag content on the border with oxidized ore. Ag grade can be up to 8 g/t and more . (Novikov et al, 2010; Scott et al, 2011).

Primary sulphide zone

Mikheevskoye deposit displays well-developed sulphide zoning as part of the dominant calcic and potassic (feldspar destructive) alteration event. Primary zone can be divided into three blocks: northern, central and southern (Bulatov et al, 2010).

Northern block**.** Northern part comprises of four zones: chalcopyrite-bornite, chalcopyrite, pyrite-chalcopyrite and pyrite. Significant amounts of magnetite could be observed only in chalcopyrite-bornite and chalcopyrite zones **(**Scott et al, 2011).

Northern block is localized mainly in upper Devonian and Low Carboniferous layer of the volcano-sedimentary rocks. This part is the metal-richest in ores. It is elongated from NE to the SW for 940 m between lines 105a and 97. Width of the block is changing between 100 and 400 m. Number of dykes is relatively small (Bulatov et al, 2010).

The richest part (in terms of Cu and Au) of Mikheevskoye deposit is chalcopyrite-bornite zone in the northern part of deposit. This is the core of the deposit and is not as structurally controlled as the broadly coincident early porphyry dyke and associated with it A-type quartz veinlets **(**Scott et al, 2011).

Although, the high-grade core cannot be distinguished on the basis of any obvious structural or secondary alteration features, it is clearly identified by its distinctive sulphide mineralogy. The concentric sulphide zones are narrower on the eastern side of the deposit than on the west, perhaps because the controlling temperature gradient was steeper on that side, rather than because of any structural control by syn-mineralization or subsequent structural modification **(**Scott et al, 2011).

***Central block*.** Central and southern blocks do not contain chalcopyrite-bornite core, however, central part includes chalcopyrite zone **(**Scott et al, 2011).

Unlike northern block, mineralization of the central block is made of granitoid dykes. Mineralization zone, which coincides by elongation and strike with a dyke belt, is represented by series of steep linear flattened plate lenses. Some lenses are steeply dipping to the east. Central block is a continuation of the Northern block. Central block is placed between lines 97 and 87 – Appendix 2, Appendix 3. It is 990 m long and 90-400 m wide (Bulatov et al, 2010).

***Southern block*.** Southern block is located in the outermost SW part of the deposit between lines 90 and 77 (Appendix 2, Appendix 3). Southern block is 1336 m long and is 100-360 m wide. The planar view of the southern block is ellipsoidal, cross section – trapezoidal (rhombic). Thickness of the block is up to 300 m. Thickness of the block is decreasing between lines 87 and 85. Thickness is not changing much with the depth. Southern block shrinks between lines 83 and 80 (Appendix 2, Appendix 3).

Unlike the northern block, granitoid dykes of the southern block developed more widely, however, mineralization here is concentrated in dykes' exocontacts in basaltoids of the Lower Carboniferous volcanogenic strata (Bulatov et al, 2010).

The pyrite-deficient parts of these sulphide-zoning patterns coincide with the highest Cu and Au grades and appear to be centered on the early porphyry dykes and A-type quartz-veinlet stock works. Although the pyrite halo is well defined, it does not have particularly high sulphide contents (3 % of volume content). This sulphide-zoning pattern also appears to have a marked influence on both the Au contents and Cu/Au ratios at Mikheevskoye deposit. Within the chalcopyrite-bornite and chalcopyrite-only zones, Au contents are elevated and the Cu correlates well with Au. In contrast, in the peripheral pyrite-chalcopyrite zone, Au contents are generally low and, where Cu contents are appreciable, there is no obvious Cu/Au correlation.

General description of the primary sulphide zone

The most important amongst three blocks mentioned above is the Northern block.

The main minerals of the primary zone are chalcopyrite and pyrite, minor - bornite, molybdenite, magnetite, pyrrhotite, sphene, ilmenite, sphalerite, galena. Molybdenum and bornite are more concentrated in volcanoclastic rocks and less - in diorites. In addition, a number of rare minerals was identified, which include faded ore, chalcocite, arsenopyrite, rutile, native gold, gold and silver tellurides.

The total content of the metallogenic minerals in sulphide ore varies between 4,7 – 7,8%. 87,8% of Cu content in the primary zone is presented by primary Cu in chalcopyrite. The total amount of oxidized and water-soluble Cu does not exceed 1,96% (Bulatov et al, 2010).

Ore bearing coefficients in stock work of the primary sulphide ore vary between 0,83 – 0,21, which means **obligatory ore assaying before extraction**. (Novikov et al, 2010)

Cu content in cores extracted during exploration works usually is sinusoidal with 2-4 picks of Cu content up to 1,5-4,0 %. In general, there is a small decrease of Cu content from the top to the bottom of the Mikheevskoye deposit. Coefficients of ore bearing are changing with depth as well. Therefore, following has to be considered:

- qualitative parameters of the primary sulphide ore are worsening in the direction of E – W;
- individual drill holes, which penetrated to the primary sulphide ore at maximum depth cross the stock work at deeper levels (below the -20 m horizon) in its central part or close to its western border;
- qualitative parameters of the primary sulphide ore in deep horizons (below -20 m horizon) are based on the individual intersections of the stock work (Novikov et al, 2010).

Parts with intensive metasomatic alterations of rocks are characterized by specific gravity of 2,8 t/m^3. Tensile strengths for compression of the rocks of this complex range between 150,4 - 158 MPa (average – 154,2 MPa). When stretched, the average value is 21,4 MPa. The angle of internal friction is 30 °, cohesion – 32,2 MPa (Novikov et al, 2010).

2. Methodology

The spatial predictive model for mineral processing plant was based on the geometallurgical zonality of the ore body and metal distribution. Metal distribution model was obtained on the basis of the hydrothermal alteration zonality of the ore body.

There were five major sequential players in the Cu production process chain: open pit, stockpile, milling department (represented by Metso), flotation department (represented by Outotec) and smelter (located in city Karabash, Chelyabinsk region, Russia). All these players were sequentially connected through their products. Open pit produces ore which is crushed and sent to the stockpile. Stockpile sends crushed ore to the milling department, where it is transformed into pulp. Pulp is sent to the flotation department where concentrate is produced. Afterwards, Cu concentrate is sent to the smelter where metal is produced (Figure 6).

Requirements to the product and feed quality were different for different stages. Quantity, Cu%, Au, top cut were currently used quality parameters for the ore; Cu% content and particle size distribution (PSD) for the pulp; and quantity Cu%, Au% content for the concentrate. However, these quality parameters were not sufficient. Hardness was an important parameter for the ore quality control. Hardness, Fe% and oxides content impacted on the flotation reagent regime of the pulp. SiO_2, Fe% and S% were influential parameters for the smelting process.

Hence, feed quality forecast system presented in this study was developed to improve the current situation with insufficient information exchange and quality control. Plenty of the production problems could be omitted or mitigated if most of the quality parameters were controlled from the early beginning (at least from the stage of feasibility study or early production). Such problems which could be mitigated were: reagent consumption planning, assuming wear of the details, energy consumption planning, design of blasting works, calibration of the control equipment (such as Courier, PSI on-line analyzers), and others.

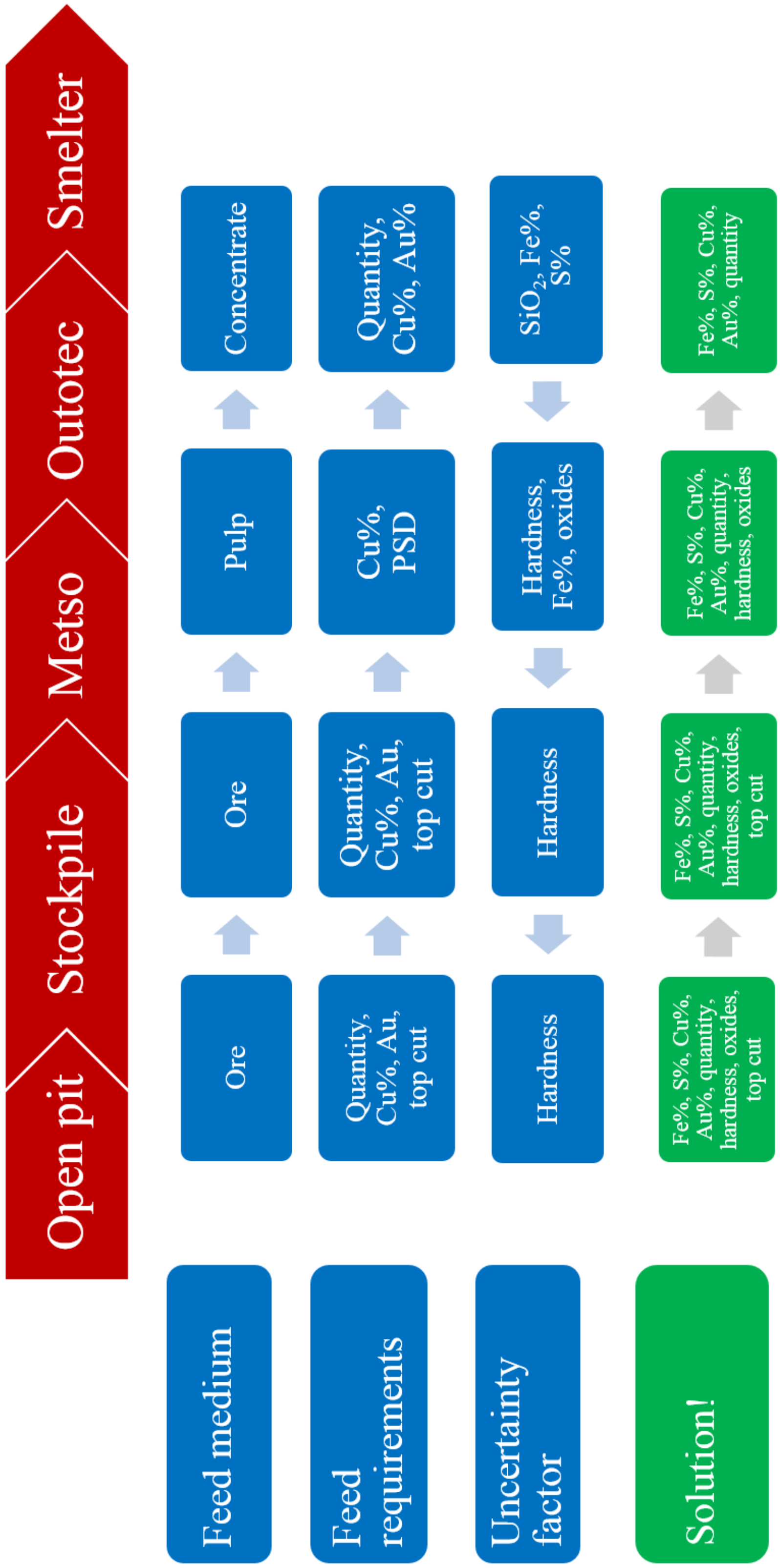

Figure 6 Problem-solution of the Cu production process from the perspective of the feed quality forecast.

2.1. Geometalurgical ore types

Definition of zonality of geometallurgical ore types had to be performed in three stages. First, theoretical geometallurgical ore types had to be specified. Later, existence of the theoretical geometallurgical ore type zones had to be confirmed with two stages: the sampling campaign and laboratory tests. Finally, the reliability of the laboratory tests of the geometallurgical ore types had to be confirmed with continuous observations of the actual mineral processing at the processing plant.

Theoretical geometallurgical ore types were specified on the basis of the geological database. Each geometallurgical ore type was described by three parameters: hardness, oxidation, magnetite presence. These parameters are listed in the Table 6 and were selected from the list of parameters available from the evaluation of the database and based on the survey conducted among Outotec, Metso and RCC engineers.

Table 6 Linkage between geology and metallurgy based on (Lamberg, 2011).

Geological/mineralogical factor		**Linkage**	**Metallurgical output**		
			Grinding	**Flotation**	**Solid-liquid separation**
Primary rock type and distribution		Hardness	X		
Ore assemblage and ore formation processes		Solubility, presence of talc, hardness	X	X	X
Alteration	Down temperature (hypogene)	Clays, hardness	X		X
	Weathering (supergene)	Solubility	Leachibility, purification		
Faulting		Clays, oxidation		X	X
Metamorphism		Clays, presence of talc, hardness	X	X	X

Existence of the geometallurgical ore types within ore body was supposed to be proved with help of a sampling campaigns. The first sampling campaign and mineralogical laboratory tests were supposed to define the base line parameters of the ore. The second sampling campaign and flotation laboratory tests were supposed to describe and confirm existence of the separate geometallurgical ore types, describe their processing regime and possibility to blend different geometallurgical ore types. However, only first sampling campaign was conducted without further flotation tests, due to financial limitations which were constraining this study.

Confirmation of the results of this study will be conducted by continuous measurements of the process parameters and laboratory tests. These measurements and laboratory tests form currently parts of the regular process, hence there are no additional costs required for the data storing and data analysis. Following equipment will be used for the process parameter measurements: “Courier 6i SL” (on-stream analyser), “PSI-300” (particle size analyser), “PSI-500” (particle size analyser), “FrothSense” (froth camera system), “Chena” (electrochemical potential analyzer) and pH meters.

2.1. Hydrothermal alteration zonality

Determination of hydrothermal alteration zonality was conducted with help of alteration data included into the geological database indirectly. It meant that hydrothermal alteration zones were defined on the basis of the mineral content. Use of hydrothermal alteration zonality in Cu content estimation allowed estimation of distribution of Cu within ore body more precisely, since Cu concentration varied between hydrothermal zones. In this case, bornite, chalcopyrite-pyrite and pyrite zones were defined and two first zones were used in Cu content modeling.

Magnetite was found to be an important vein and alteration mineral in the high-grade core of some gold-rich deposits, and could locally comprise up to 10 w-% (Holliday and Cooke, 2007**).** Analysis of the geological database had revealed some correlation between Au content and magnetite distribution (mean value of Au content within the magnetite zone was 28% higher than ouside the magnetite zone, shown in Table 13), thus Au content was modelled separately within and outside magnetite zone.

2.2. Geostatistical modelling

Estimation of the metal distribution was done with use of ordinary Kriging. However, the same calculations were repeated with the inverse distance method in order to confirm obtained results. More attention would need to be paid to the variogramms and geostatistical modeling in the future research involving specialists in this area. Otherwise, obtained results wouldn't be considered reliable enough. Hardness distribution was estimated with the use of the nearest neighbor method.

2.3. Open pit

The largest economically feasible open pit and its intermediate stages were crucial for the feed quality forecast. They supplied the information on the total amount of all the geometallurgical ore types and metal content which could be extracted during certain time ranges. Open pit optimization was performed with the use of special optimization tool available in Surpac 6.3. All relevant economic data (used for the optimization) had been corrected for the inflation and in accordance with the market situation. The corrected data included metal prices, the average processing costs and mining costs.

Optimized open pit model was an approximate model of the future real pit. The real pit might end up with different volume (up to ±15%) than the volume of the optimized pit. Therefore, detailed design of the open pit was made on the basis of the optimized open pit model. Some simplifications were made in the designed pit comperatively to the open pit design developed by RCC: only one exit ramp was implemented instead of two; no double or triple benches had been used. None of these changes had any significant influence on the final result.

2.4. Mining parameters

From the mining perspective it is important to notice that ore in Mikheevskoye deposit occurs at depth of 2-10 m. Terrain in the exploitation region is rather flat. Mineralization of the ore body has a large extent and metal content is low.

(Scott et al, 2011) proposed the following open pit design parameters (Table 7). Same parameters have also been used in this study.

Table 7 Recommended pit parameters at the limiting contour (Scott et al, 2011).

Number and name of the zone	**Horizon**	**Bench slope angle, degree**	**Safety berm width, m**
1. Top soil and clay	Surface – 24 m	40	8
2. Weathering crust	24 m – 50 m	50	8

Number and name of the zone	Horizon	Bench slope angle, degree	Safety berm width, m
3. Transition zone	50 m – 80 m 80 m –140 m	60 65	8 8
4. Zone of unaltered rocks	140 m –300 m	70	8
Slope of the ramp = 10 %, width of the ramp = 30 m, berm width = 10 m (up to horizon 250), benches at the horizons 250-80 to be 15 m, safety bench – 8 m wide.			

2.5. Mineral processing

2.5.1. Oxidized ore

Previously it was stated that oxidesed ore has to be treated by means of heap leaching (IAC Mineral /ИФЦ Минерал, 2011). However, in (Beloshapkov and Popov. 2012), it is stated that investigation in heap leaching resulted in low recoveries with extremely high sulfuric acid requirements. In situ leaching also was rejected due to high risk of loss of pregnant leach solution in underground horizons. So, there would be no future processing for the oxidized ore.

2.5.2. Transitional ores

"Mekhanobr Engineering" (St. Petersburg) has conducted flotability tests with transitional ore in 2005. Results have revealed that bulk concentrate after the fourth cleaning recovers: 50,88% Cu (grade was 25.68%), 33,87% Au (grade was 2,86 g/t), 38,44% Mo (grade was 0,174%). It was concluded that mixed ores could be flotated, however, recoveries are very low. Therefore, processing of this ore type separately will not be profitable (Bulatov et al, 2010).

2.5.3. Primary sulphide ores

Flotation tests of 164 kg of the primary sulphide ore samples were conducted.

Following recoveries were obtained: 88,36% Cu (grade 3,26%), 83,01% Au (grade 0,82 g/t), 67,97% Mo (grade 0,024%). Cu recovery in the concentrate of the second cleaning was 81,38-82,02% Cu (grade 20-19,36%), 61,57% Au (grade 3,5 g/t), 51,5% Mo (grade 0,127 g/t) (Bulatov et al, 2010). Outotec has also conducted flotation tests in 2008 with following results (Table 8).

Table 8 Mineral processing parameters of the primary sulphide ore in 2008, conducted by "Outotec Minerals Oy" (Finland) (Bulatov et al, 2010).

Products	**Mass pull, %**	**Grade, % (g/t)**				**Recovery, %**			
		Cu	**Mo**	**Au**	**Ag**	**Cu**	**Mo**	**Au**	**Ag**
Cu concentrate	1,7971	20,03	0,045	(6,72)	(20,55)	79,99	20,00	58,0	50,0
Mo concentrate	0,0029	2,00	48,00	-	-	0,01	35,00	-	-
Cu-Mo concentrate	1,80	20,00	0,122	(6,72)	(20,55)	80,00	55,00	58,0	50,0
Tails	98,20	0,09	0,002	(0,094)	(0,38)	20,00	45,00	42,0	50,0
Ore	100,0	0,45	0,004	(0,22)	(0,74)	100,0	100,0	100,0	100,0

The general mineral processing flow chart of the Mikheevskoye is given in Appendix 4.

2.6. Mine scheduling

Investigation of the optimal feed quality plan for the processing plant was done through the mine scheduling. In order to create mine schedule, the intermediate open pits were designed. Each intermediate open pit corresponded to the stage of mine development in the future. Ore body blocks which appeared between intermediate open pit models were to be extracted in the time range to which the upper and lower open pit models corresponded. The final intermediate open pit used in the mine scheduling corresponded to 5,5 year of the mine's full capacity operation.

Use of the geometallurgical ore type zonality in the mine scheduling also would need to provide the highest possible metal content in the plant feed. Hence, two separate mine schedules were created. One mine schedule was based on feed quality totally dependent on the geometallurgical ore type. Another schedule was based on the preferential extraction the highest Cu content ore first.

3. Data collection

Three major sources of information have been used for data collection: databases, sampling and a survey. Databases were obtained from MiGOK and Outotec. One sampling campaign has been conducted on 02-07.10.2013 and its results are given below. Survey was conducted among Outotec, MGOK and Metso.

Sampling campaign was conducted in order to provide the base line data on the ore quality. Survey was aimed to confirm if the selected theoretical geometallurgical ore types were correct. Theoretical geometallurgical ore types were based on the combination three parameters: oxides content, iron content (magnetite) and hardness.

3.1. Sampling

3.1.1. Minimum sample weight and the fundamental error

It is crucial to define sampling parameters before the sampling campaign will start. One of such parameters is minimum sample weight. There are many methods to define the minimum sample weight (Tomanec and Milovanovic, 2005). However, the following formula was chosen for defining minimum sample weight (Holmes, 2004):

$$m_S = \frac{c \cdot l \cdot f \cdot g \cdot d^3 \cdot a^2}{\sigma_{FE}^2} \tag{1}$$

where:

- c is the mineralogical composition factor. Preliminary geological research has shown that the fractional concentration of the component of interest a = 0,41%. Density of the gangue particlesρ_2 = 2,6 g/cm^3, density of the particles ρ_1 = 4,2 g/cm^3 (for the chalcopyrite). Therefore, the mineralogical composition factor is equal:

$$c = \frac{(1-a) \cdot \left((1-a) \cdot \rho_1 + \rho_2 \cdot a\right)}{a} = \frac{(1-0{,}41\%.) \cdot \left((1-0{,}41\%.) \cdot 4{,}2 + 2{,}6\ \cdot 0{,}41\%.\right)}{0{,}41\%.} = 1018{,}60 \tag{2}$$

- l is a liberation factor. The liberation factor was defined on the basis of the nominal top size $d_l = 150\ \mu$ and top cut of the feed d = 250 mm

$$l = \sqrt{\frac{d_l}{d}} = \sqrt{\frac{0{,}015\ \text{cm}}{25\ \text{cm}}} = 0{,}024 \tag{3}$$

- *f* is particle shape factor, which can usually be taken to be 0,5;
- g is a size range factor, usually between 0,25 and 1,0 g is defined on the basis of the nominal top size of the material (*d*) to the lower size (about 5% undersize) (*d'*) (*Table 9*).

Table 9 Size range factor selection.

Large size range	*d / d' >4*, [mm/mm]	$g = 0{,}25$
Medium size range	*2 ≤ d / d' ≤ 4*, [mm/mm]	$g = 0{,}50$
Small size range	*d / d' <2*, [mm/mm]	$g = 0{,}75$
Uniform size	*d / d' =1*, [mm/mm]	$g = 1{,}00$

- a is a fractional concentration of the component of interest,%
- σ_{FE}^2 is the fundamental error as a fractional concentration.

The technical support of the sampling campaign (using only labor force, limited access to the equipment) was a limitation, which allowed to collect only m_S = 2,4 ± 15% t of samples. Therefore:

$$\begin{aligned} 400\ kg &= \frac{c \cdot l \cdot f \cdot g \cdot d^3 \cdot a^2}{\sigma_{FE}^2} = \frac{1018{,}60 \cdot 0{,}024 \cdot 0{,}5 \cdot 0{,}25 \cdot 25^3 \cdot 0{,}41\%^2}{\sigma_{FE}^2} \\ &= \frac{0{,}82}{\sigma_{FE}^2} \rightarrow \sigma_{FE}^2 = \frac{0{,}82}{2400} = 0{,}034\% \end{aligned} \tag{4}$$

Then, the fundamental error for the chalcopyrite content in a sample is:

$$\sigma_{FE}^{CuFeS_2} = \sqrt{0{,}034\%} = 1{,}847\% \tag{5}$$

And fundamental error for the Cu content is (Cu is approximately 1/3 of chalcopyrite grain by weight):

$$\sigma_{FE}^{Cu} = \frac{\sigma_{FE}^{CuFeS_2}}{3} = \frac{1{,}847\%}{3} = 0{,}616\% \tag{6}$$

Since, $\sigma_{FE}^{Cu} > a$, it could be concluded that sample might be not very representative, however, the representativeness of the sample could be confirmed by metallurgical analysis. Then, Cu content has to be above 0,2%.

3.1.2. Sampling area

Samples were taken from 4 different areas: ore stockpile (36), ore stockpile (33) and two different places in the open pit (1). One area in an open pit, was a horizon 240 of the open pit's central part (just few days after blasting, which was made on 30.09.2013). Another area was in the southern part of the open pit; see Figure 7 for more details.

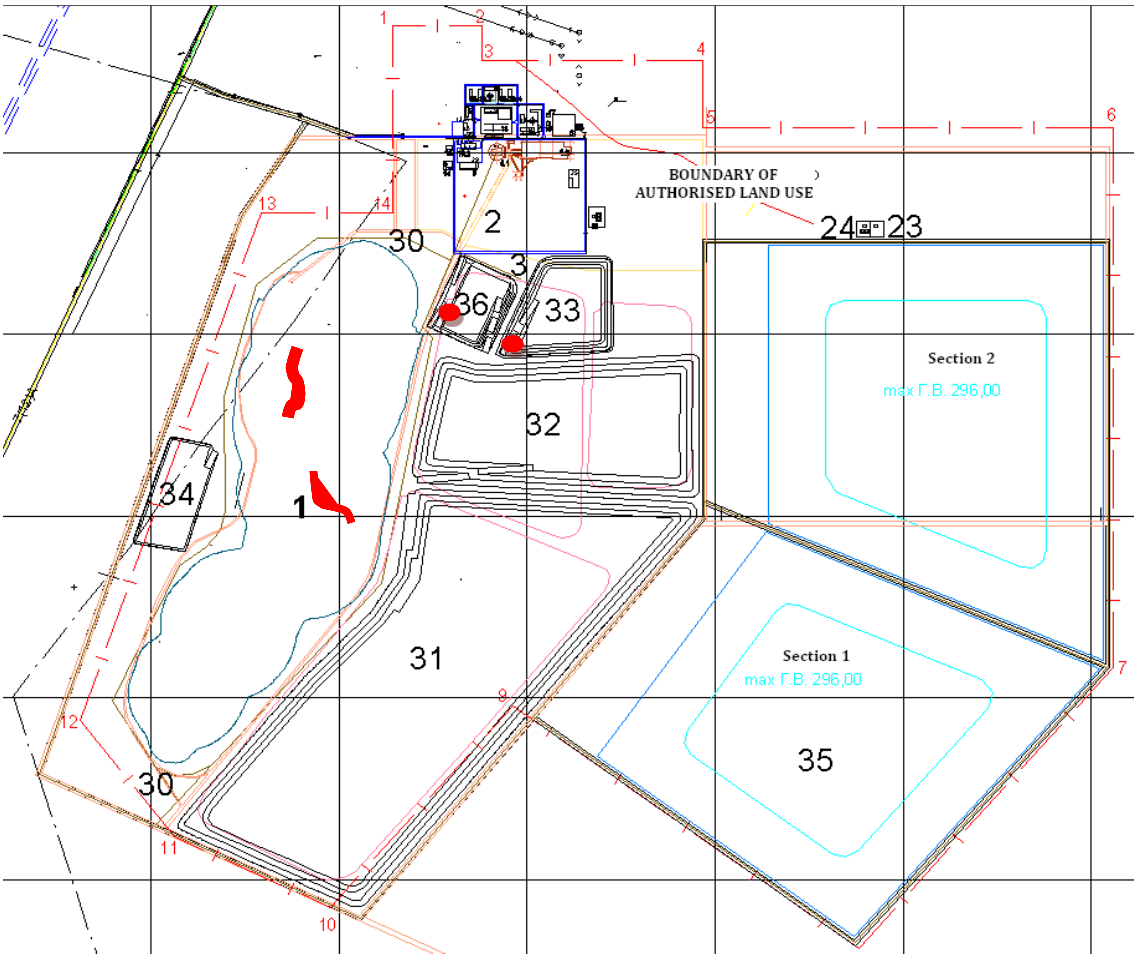

Figure 7 Mikheevskoye GOK (Sampling areas are marked as red).

The sampling area was divided into zones where samples were taken only from the zones where it was safe to collect samples. Red arrows in (Figure 8) represent sampling points which are too close to the edge of the open pit. Green arrows represent sampling points which are located far enough from the edge of the slope. Sampling points with oxidized ore were avoided (Figure 8).

Typical sample extraction method from the stationary mass (according to Russian rule of thumb) presumes digging a small hole in a heap (0,2-0,4 m) and further removing of the sample with a help of shovel/scoop.

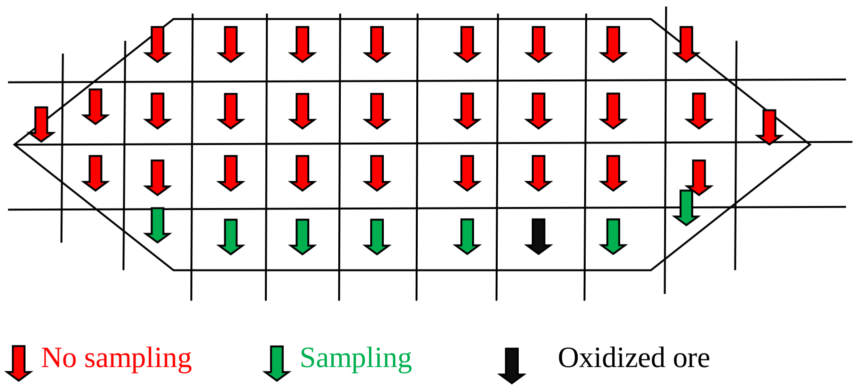

Figure 8 Sampling area.

Due to lack of equipment support, sample extraction was made on the basis of following pattern Figure 9. When it was possible, samples were taken 1 m away from the edge of the heap.

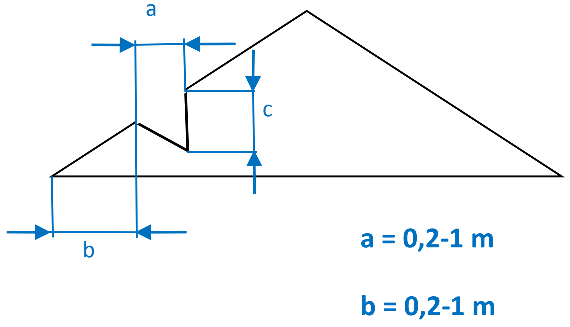

Figure 9 Sample extraction from the stationary mass.

As a result of the sampling campaign, 2,4 t of primary samples were collected and 600 kg of secondary samples were selected after mixing and separation.

3.1.3. Sample preparation

Parameters of the sampling preparations:

- The total amount of the secondary sample is 600 kg where 300 kg will be used for the laboratory tests and 300 kg will be stored in the Outotec office before the test results won't be revealed.
- Samples' quality has to correspond to the future feed.
- The maximum particle size has not exceeded 250 mm.
- The secondary 600 kg of the sample was taking after careful mixing and separation. The whole procedure of mixing was repeated twice.

The intermediate storing was made in the temporary empty lime storage. The floor in the storage was made out of concrete. Before the primary sample was dumped, the floor was carefully cleaned. There was no special cover used to put under the primary sample. After the primary sample was dumped, the storage area was surrounded by the rope, in order to protect the pile.

The ore mixing was made manually with a use of shovels. Mixing and separation was conducted according to the following scheme in a three phases (Figure 10):

1. Pile (A1), was divided into 4 piles (B1, B2, B3, B4);
2. Two the most remote piles were mixed (C1,C2),. So, B1 and B4 piles were put together in C1, and B2 and B3 were put into C2;
3. Two newly created piles (C1,C2) were mixed into (D1);
4. 1-3 were repeated: D1→ E1, E2, E3, E4 → F1, F2 → G1;
5. Pile (G1) was divided into 3 piles (H1, H2, H3);
6. One pile (H1 or H2 or H3) was packed into the bag HX.

As a result of the sampling campaign, 2,4 t of primary samples were collected and 600 kg of secondary samples were selected after mixing and separation.

The average metal content in collected samples was slightly above the cut-off grade of 0,2% Cu (Table 10), which showed poor knowledge of the metal content and metal distribution in the already excavated ore. This result proved that more communication had to be established between RCC geologists, open pit engineer and plant's production planners. Otherwise, feed quality might be not sufficient and uneven.

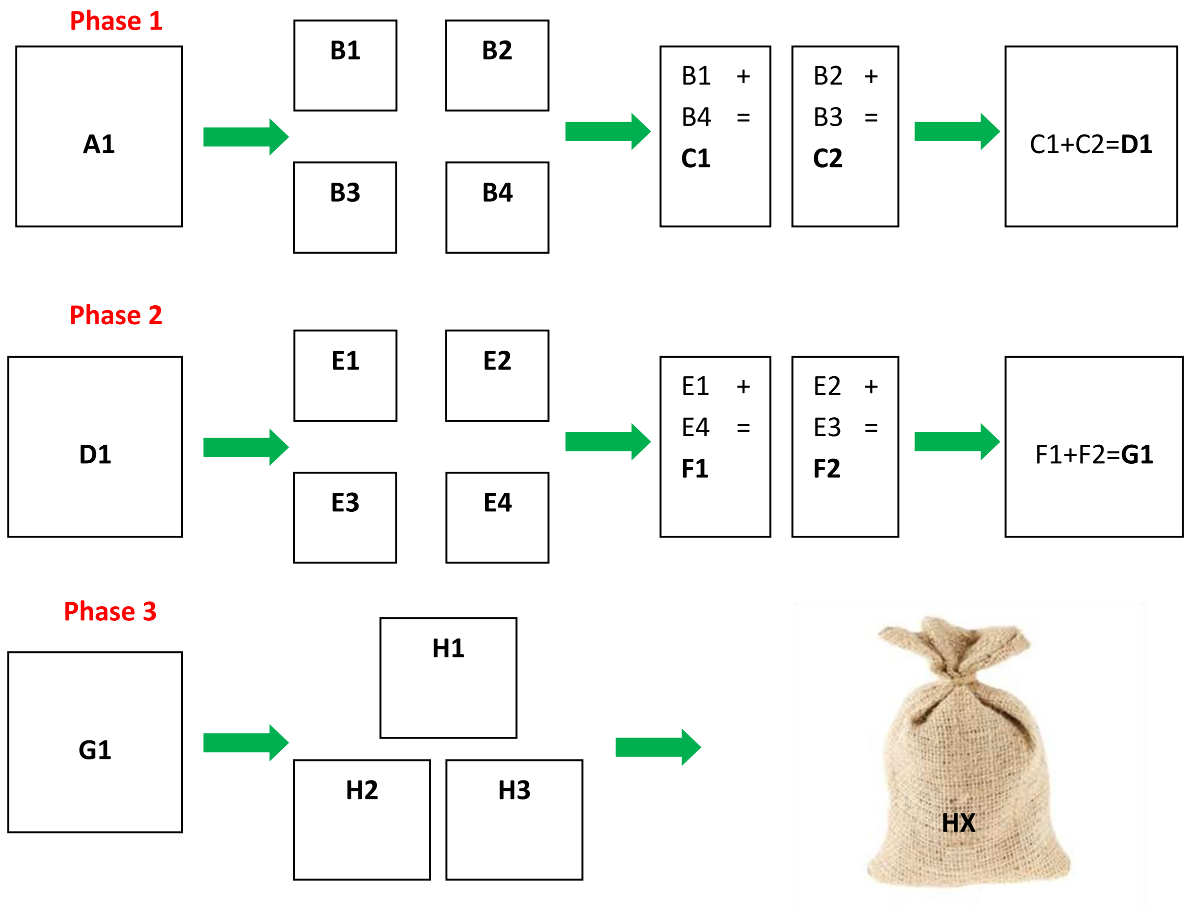

Figure 10 Sample preparation: ore mixing

Table 10 Results of the laboratory tests.

№	Laboratory sample code	Sample code	Cu general,%		Ag, g/t		Au, g/t	
			main	control	main	Control	main	control
1	472-13-001	O1	0,207	0,213	1,8	1,8	0,1	0,1
2	472-13-002	O2	0,218	0,227	1,0	1,0	0,1	0,1
3	472-13-003	O3	0,093	0,092	<1,0	<1,0	<0,1	0,1
4	472-13-004	O4	0,388	0,405	1,0	1,0	0,1	0,1
5	472-13-005	O5	0,183	0,186	1,0	1,0	0,1	0,1

Since Cu grades of the collected samples (Table 10) were lower than expected, sampling campaign has to be repeated. The expected Cu grade at current production stage is 0,6-1,0%. Sampling campaign had to be planned in cooperation with mine surveyor in accordance with geometallurgical ore type zonality and the most recent mine schedule. Sampling campaign had to be conducted under strict coordination of the chief geologist and surveyor.

It is also suggested to conduct a second sampling campaign. Second sampling campaign will be aimed to confirm presence of separate geometallurgical ore types within ore body.

3.2. Questionnaire study

More than 15 questionnaires were distributed among MGOK, Outotec, Metso employees and seven (7) fully filled in questionnaires were received back.

Survey aimed to figure out importance of the plant's feed parameters: iron content - in this case equal to magnetite content, quartz content - corresponds to hardness, sulphur content, Cu, Au and oxides content. Questions were related to the different parts of the beneficiation process: reagent and electricity consumption, Cu and Au recoveries, maintenance of the most important equipment (such as crusher, mills, Isa-mill, flotation cells, filters, pumps). Respondents were requested to answer questions on the scale from 1 to 6, where the least important was 1 and the most important was 6 (Table 11).

Table 11 Mean values of the importance of the different components' content in the plant's feed according to the survey respondents (on the scale between one and six, where "one" is the least important and "six" is the most important)

		Iron	**QTZ**	**Sulphur**	**Cu**	**Au**	**Oxides**
Consumption	**Reagents**	3,14	4,29	4,86	5,86	4,17	4,14
	Electricity	4,00	5,29	4,33	4,14	3,5	3,00
	Mean	***3,57***	***4,79***	***4,60***	***5,00***	***3,83***	***3,57***
Maintenance	**Crushing**	4,83	5,43	4,50	3,67	3,17	3,17
	Milling	5,00	5,71	4,71	4,00	2,8	4,00
	Flotation	4,86	5,86	4,67	4,33	3,5	3,67
	Isa-mill	5,43	5,57	4,71	4,57	3,67	4,14
	Filtration	5,14	5,43	4,50	5,14	4,67	4,71
	Pumps	4,83	5,83	4,50	4,33	3,67	4,17
	Mean	***5,02***	***5,64***	***4,60***	***4,34***	***3,58***	***3,98***
Recovery	**Cu**	4,43	4,00	5,00	6,00	3,43	4,71
	Au	3,83	4,17	4,17	4,17	6,00	3,50
	Mean	***4,13***	***4,08***	***4,58***	***5,08***	***4,71***	***4,11***
Total average		***4,24***	***4,84***	***4,59***	***4,81***	***4,04***	***3,88***

Data in the Table 11 has revealed that Cu content and quartz content (or hardness) are of the utmost importance for the mineral processing and equipment maintenance. Sulphur and iron have less impact on the process; Au and oxides content is the least important.

However, Cu, quartz and sulphur content are the most important for the reagent and electricity consumption. Quartz and iron content have the highest influence on the maintenance cost. Oxides and sulphur (besides Cu and Au content) content play the most important role in defining recovery.

Additional question was asked about cost dependence between concentrate production cost and iron, quartz, sulphur, Cu, Au and oxide content in the plant's feed. The question was aimed to find out what will be the change of the concentrate cost in % if any of the variable parameters (iron, quartz, sulphur, Cu, Au and oxides content) will rise by 1%.

The last question concerned the acceptable range of iron, quartz, sulphur, Cu, Au and oxides content in the feed (Table 12).

Table 12 Cost, energy and reagent consumption changes (%) depending on the feed quality and content ranges of the feed components if any of the variable parameters (iron, quartz, sulphur, Cu, Au and oxides content) will rise by 1%.

	Cost change			**Content**		
	Costs, %	**Reagents, %**	**Electricity, %**	**Min, %**	**Mean, %**	**Max, %**
Iron	3,25	2,33	4,00	0,50	0,90	2,00
QTZ	4,60	7,40	4,60	0,00	61,00	95,00
Sulphur	2,50	1,75	5,20	0,60	0,95	1,75
Cu	-4,00	-2,00	2,50	0,42	0,65	1,14
Au	-1,00	-1,25	2,25	0,06	0,26	0,81
Oxides	2,17	1,33	4,33	0,00	7,50	10,00

Table 12 reveals high importance of the quartz content and thus, significance of the hardness, which is Bond index in case of mineral processing, for the cost and reagent consumption. Cu and iron content have significant impact on cost of concentrate production. Electricity consumption is dependent on sulphur, quartz and oxides content.
The range of change for some components in the feed shows misunderstanding of the ore quality parameter among respondents. Results obtained from the survey are bit higher

than expected, since real average Cu content from geological data was estimated close to 0,4% (while survey shows 0,65%) and Au content close to 0,1 g/t (survey shows 0,26 g/t).

Results obtained from the survey have limited usage since they reflect only process engineers' and maintenance engineers' opinions. However, the average values show that theoretical parameters for geometallurgical ore types (oxides content, iron content (magnetite) and hardness) were selected correctly.

3.3. Drilling database

Data mining and documentation review has shown that there are several sets of data which are used as primary sources of geological information by different stakeholders. Four of the most recent sets of information are the following:

- Geological data base developed by Celtic and later used by Outotec, provided by Markku Meriläinen (Outotec).
- Geological data base developed by Ural-VCM by Belyashov Ilya, provided by geological survey of MGOK.
- Geological data base developed by Ural-VCM, provided by geological survey of MGOK.
- Geological data base (called as "Set_2010_comb") provided by Markku Meriläinen and Pekka Loven. This data base is a combination of the above mentioned data bases. Several errors were detected and corrected manually in the database. Drilling database Set_2010_comb has the following structure: 21479 Cu assays, 11552 Au assays, 1730 density assays, 608 hole identification numbers. The spatial extensions of the drill holes are given in the Appendix 5.

This database is mainly concerned with rock types. However, good logging database has to present alterations but not just rock types. This is true for the majority of Cu-porphyry deposits in the world. The rock type and alteration were mixed in the current database and it has to be rellogged according to alteration but not rock type. So, more drilling has to be performed in the future. Nevertheless, there was no possibility for the additional drilling for further research at the Mikheevskoye project during the research period of 2013-2014.

3.3.1. Assaying

Twelve different laboratories have participated in assaying drill cores at Mikheevskoye project with employing different methods: mainly X-ray fluorescence analysis (XRF), atomic absorption (AA), x-ray spectrographic (PCA) and chemical analysis, with including internal (4.8%) and external (9.1%) control assays, e.g.:

- 1987, PGO "Uralgeologia» (Sverdlovsk, Russia) laboratory has conducted drill cores assaying;
- 1999, OAO "Uralmiekhanobr" (Yekaterinburg, Russia), KHD "Humboldt Wedag" (Germany);
- 2005 – 2008, ZAO "Polymetal engineering" (St. Petersburg), OAO IRGIREDMET (Irkutsk, Russia), Anime Global Limited (Perth, Australia), "Mekhanobr Engineering" (St. Petersburg).

More than 25 000 samples were assayed for Cu content, 15 000 (including > 750 composite samples) – Au, 4 000 (including > 800 composite samples) – Ag and 5 000 (including > 800 composite samples) – Mo **(**Scott et al, 2011).

4. Modelling

4.1. Solid modeling

Solid models in geological software are used for constraining initial data or constraining the output data during the metal distribution/content estimation. Three solid types have been created:

1. Ore type (Figure 11):
   a. solid which envelopes oxidized ore type ;
   b. solid which envelopes transitional (mixed) ore type;
   c. solid which envelopes primary ore type.
2. Grades and content:
   a. solids were created for three (3) different cut-offs:
      i. 0,2% Cu grade (Appendix 7);
      ii. 0,3% Cu grade (Appendix 8);
      iii. 0,4% Cu grade (Appendix 9);
   b. solid which envelopes magnetite-rich area (Appendix 10).
3. Porphyry zonality was supposed to be performed based on the hydrothermal alteration zonality. However, sulphide zonality was used because hydrothermal alteration data in database is poor:
   a. bornite-rich zone (Appendix 10);
   b. chalcopyrite-rich zone (coincides entirely with “0,2% Cu grade” solid in (Appendix 7),
   c. pyrite-rich zone.

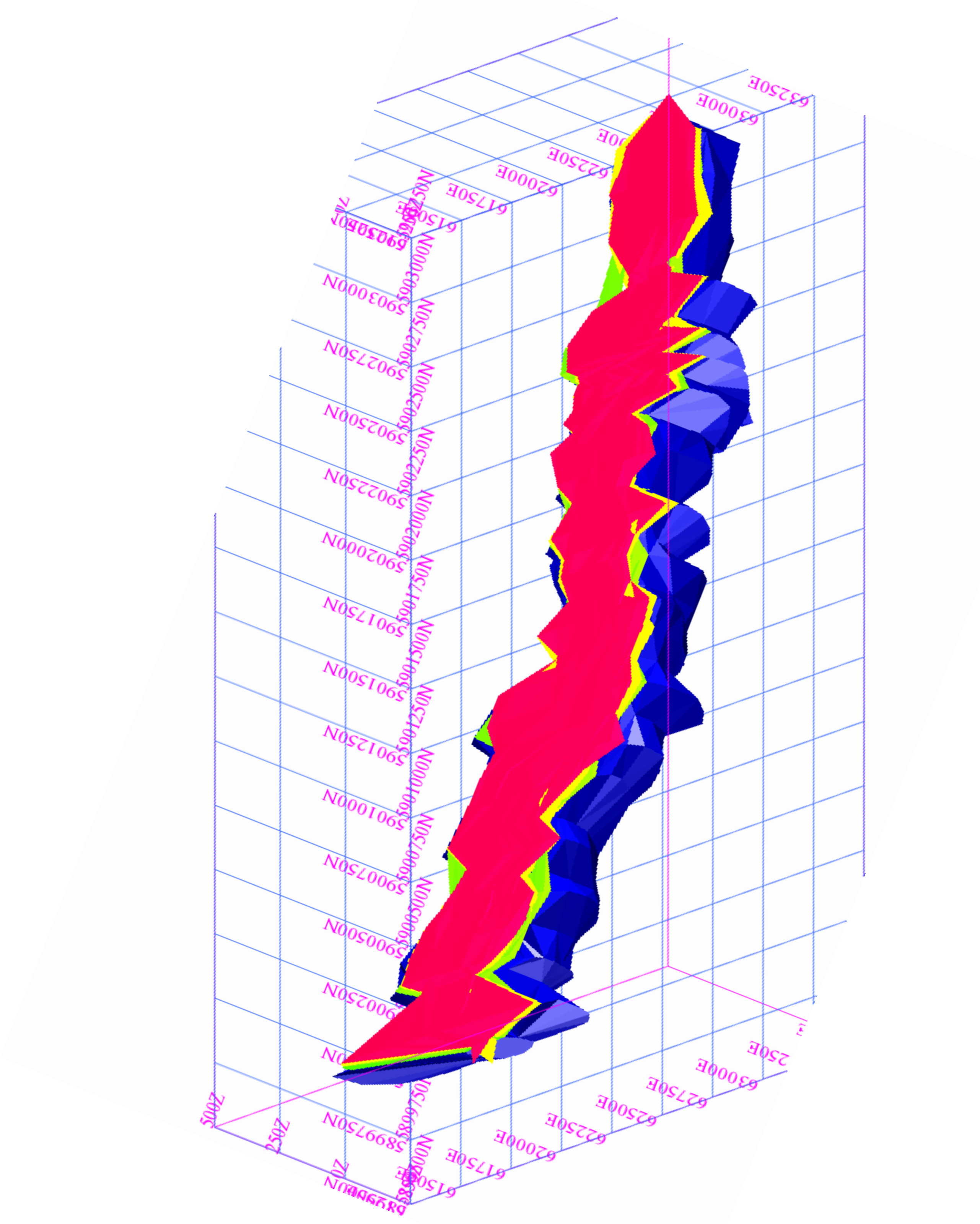

Figure 11 Weathered layers in orebody (Surpac 6.3, 3D).

4.2. Block modeling

Block model was created for the purpose of metal values estimation and had dimensions presented in Appendix 6.

Block model includes following attributes:

- "**au**" is gold content, g/t;
- "**cu**" is copper content, %;
- "**hardness**" is hardness of the rock (hard – 1, very hard – 2, extremely hard – 3, other - 0);
- "**magnetite**" is magnetite-rich zone content;
- "**material_name**" is geometallurgical zone, 1, 2 … 12;
- "**nsr**" is net smelter return;
- "**sg**" is specific gravity, t/m^3;
- "**sulphidization**" where pyrite (code - 0), bornite (1) and chalcopyrite (2) zones are recorded;
- "**weathering**" where (1) oxidized ore (including all waste rock), (2) mixed ore and (3) primary ore are recorded;
- other fields do not contain relevant information.

4.3. Composites

All composites were created on the same basis: composite length – 2 m, composite length determined by "best fit", with minimum 50% of samples included. All composites were constrained by the 0,2 grade solid (within chalcopyrite zone) and by solid which envelopes primary and mixed ore zones.

Three groups of composites have been created: Cu composite, Au composite and hardness composite. Basic statistics analysis of the composites is given in the Table 13.

Table 13 Basic statistics of the composites (ungrouped data).

File	Au	Magnetite in	Magnetite out	Cu	Bornite in	Bornite Out	Hardness
String range	All	All	All	All	All	All	All
Variable	Au			cu			hardness
Upper cut	999,00	999,00	999,00	999,00	999,00	999,00	999
Number of samples	13,79	5,46	8,33	13,79	2,87	10,92	6,03

File	Au	Magnetite in	Magnetite out	Cu	Bornite in	Bornite Out	Hardness
Minimum value	-	-	-	-	0,01	-	1
Maximum value	1,44	1,44	1,08	4,52	4,52	2,23	3
Mean	0,06	0,07	0,06	0,41	0,60	0,36	2
Median	0,02	0,03	0,02	0,34	0,49	0,31	2
Variance	0,01	0,01	0,01	0,09	0,18	0,06	0
Standard Deviation	0,10	0,11	0,09	0,31	0,42	0,24	1
Coefficient of variation	1,55	1,49	1,59	0,75	0,71	0,67	0

4.4. Outliers

Outliers are data which does not fit within data range and thus has to be removed. None of the value could be considered as an outlier due to relatively small maximum values of the metal content (Table 13).

4.5. Variogram

Data value changes over distance and direction could be understood through the variogram modeling. Variograms were modeled for three variables, namely Cu, Au and hardness (Table 14). Visualization of the variograms is given in the (Appendix 11).

Table 14 Variogram parameters.

Variable	Cu	Au		Hardness
Ellipsoid plunge	- 57,55	- 77,00		- 77,00
Ellipsoid bearing	326,71	258,00		258,00
Ellipsoid dip	- 30,00	40,00		- 50,00
major:semi-major	1,69	3,35		2,32
major:minor	2,95	5,78		8,07
Current variogram model parameters				
Model Type	Spherical	Spherical		Spherical
Nugget	0,1085	0,1427		0
Structure	1	1	2	1
Sill	0,89	0,05	0,81	1,00
Range	77,34	39,96	204,31	208,32

4.6. Values estimation

There are three types of values presented in block model: assigned, calculated and estimated.

1. Specific gravity was an assigned value;
2. Cu, Au were interpolated within cut-off grade and ore zone constrains;
3. NSR – net smelter return was a calculated attribute;
4. Hardness was estimated by nearest neighbor method.

4.6.1. Specific gravity estimation.

Based on the data given in the "3.1.5. Mineralization" and on the personal discussions with the Mikheevsoye GOK geologists, specific gravity of 2,81 t/m^3 was assigned for primary sulphide ore and 2,21 t/m^3 for the mixed, waste and oxidized ores in the block model.

4.6.2. Cu, Au estimation

Three methods were used for Cu and Au estimation: ordinary krigging when alteration zonality was applied (further refered as ok_zonned), ordinary krigging when the whole ore body is treated as unzoned (further refered as ok_unzonned) and inverse distance method when the whole ore body is treated as unzoned (further refered as id2_unzonned). Search parameters used for interpolation are given in Table 14. Additional interpolation parameters are following for all the methods:

- Max search distance of major axis: 204 m
- Max vertical search distance: 999 m
- Maximum number of informing samples: 15 m
- Minimum number of informing samples: 3 m

4.6.3. Model validation

Obtained models were analyzed by three methods: grade versus length, grade versus depth and grade versus tonnage. These analyzes have shown that there is no big difference between applied estimation methods.
Following indications were used:

- "ok_zonned" represents data obtained from the ordinary krigging applying alteration zonality,
- "ok_unzonned" represents data obtained from the ordinary krigging without applying alteration zonality,
- "id2_unzonned" represents data obtained from the inverse distance method without applying alteration zonality,

- "model" represents data obtained from the composits (raw data).

Grade versus length

Grade versus length provides an information on how the average metal grade changes along the coordinate Y. Grade versus length, where an average grade were taken between Y coordinates, is given in (Table 15).

Table 15 Locations of point Y.

Minimum Y	Maximum Y	Point Y	Minimum Y	Maximum Y	Point Y
5 899 580	5 899 922	1	5 901 290	5 901 632	6
5 899 922	5 900 264	2	5 901 632	5 901 974	7
5 900 264	5 900 606	3	5 901 974	5 902 316	8
5 900 606	5 900 948	4	5 902 316	5 902 658	9
5 900 948	5 901 290	5	5 902 658	5 903 000	10

Grade versus length analysis has confirmed that northern part of the ore body has in general higher Cu grade then southern has (Figure 12).

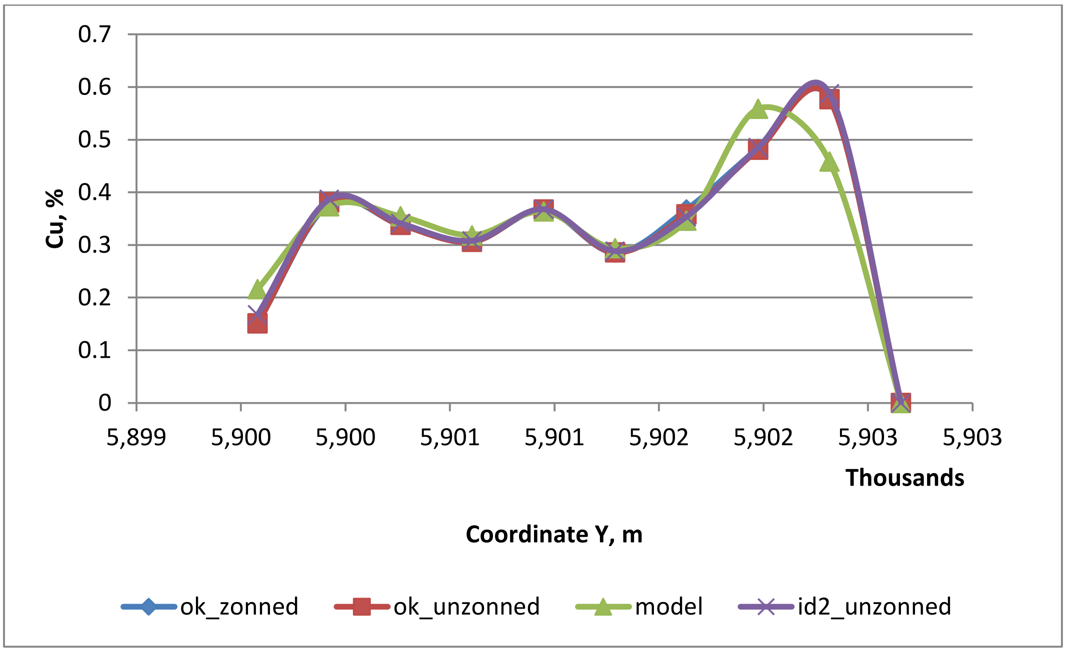

Figure 12 Cu grade versus coordinate Y (length).

Au grade versus length analysis has revealed that there are three areas of high gold concentration within ore body (Figure 13)

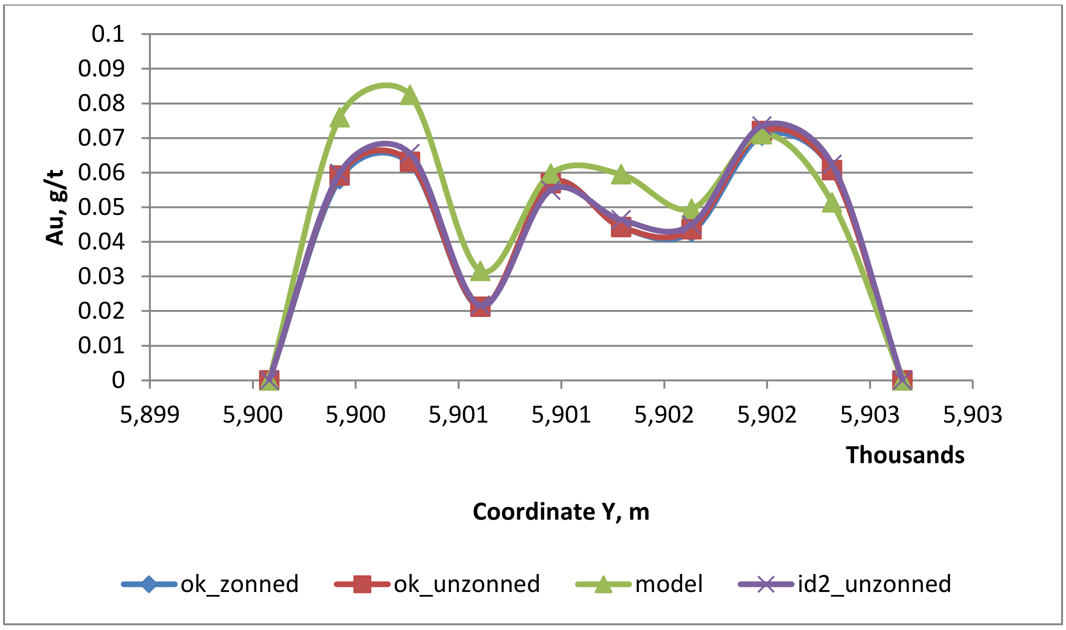

Figure 13 Au grade versus coordinate Y (length).

All three approaches give roughly the same level of accuracy. The difference between estimated values taken from the block model and values taken from the composites are very close to each other and in average do not exceed 0,055% for Cu and 0.015 for Au.

Grade versus depth

Grade versus length provides information on how the average metal grade changes along the coordinate Z. Grade versus length shows how the average grade changes along the coordinate Y. Grade versus depth, where an average grade were taken between Z coordinates, is given in (Table 16).

Table 16 Locations of point Z.

Minimum Z	Maximum Z	Point Z
-196	-130	1
-130	-63	2
-63	4	3
4	71	4
71	137	5
137	204	6
204	270	7

Grade versus depth analysis has shown completely different results for Cu and Au. While Cu tends to be higher in the upper and middle part of the ore body (Figure 14), Au content is definitely increasing with depth (Figure 15)

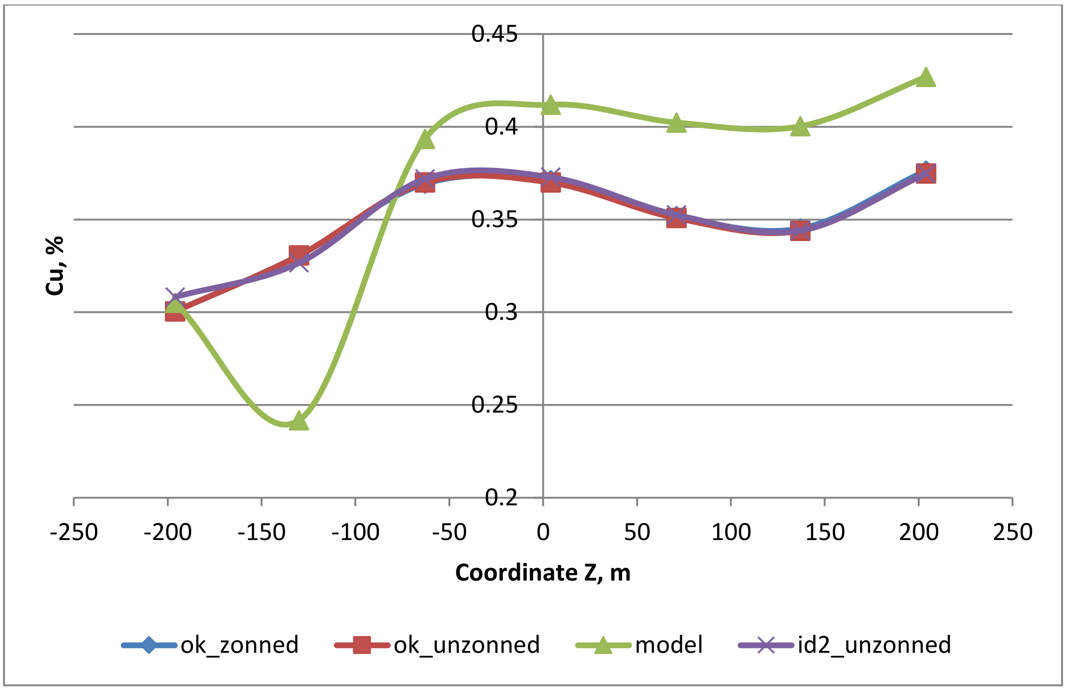

Figure 14 Cu grade versus coordinate Z (depth).

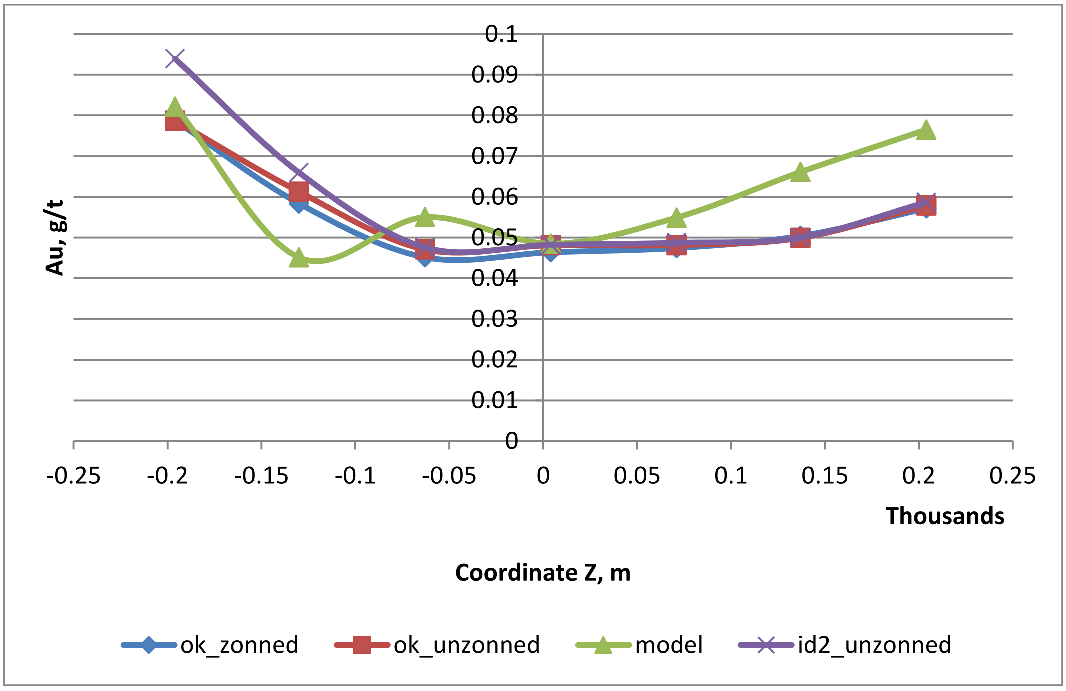

Figure 15 Au grade versus coordinate Z (depth).

All three approaches give roughly the same results. The difference between estimated values taken from the block model and values taken from the composites are very close to each other and in average do not exceed 0,07% for Cu and 0,017 for Au.

Grade versus tonnage

Grade versus tonnage analysis shows how much ore in the ore body has certain metal content. This information helps to plan the feed quality to the plant especially if ore blending is assumed (Figure 16, Figure 17).

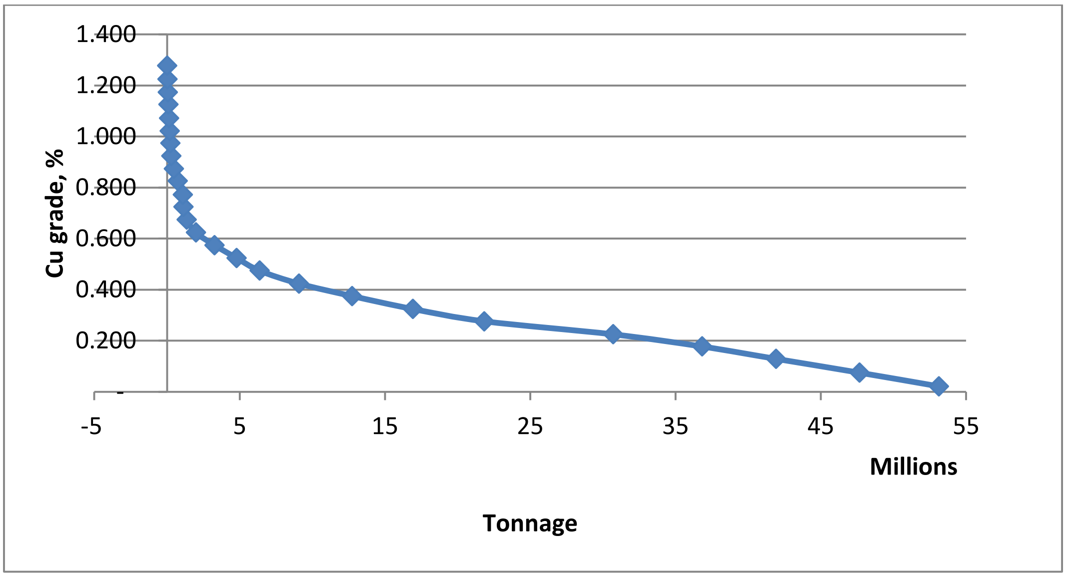

Figure 16 Cu grade (%) versus ore tonnage, Mt (ordinary kriging, zoning applied).

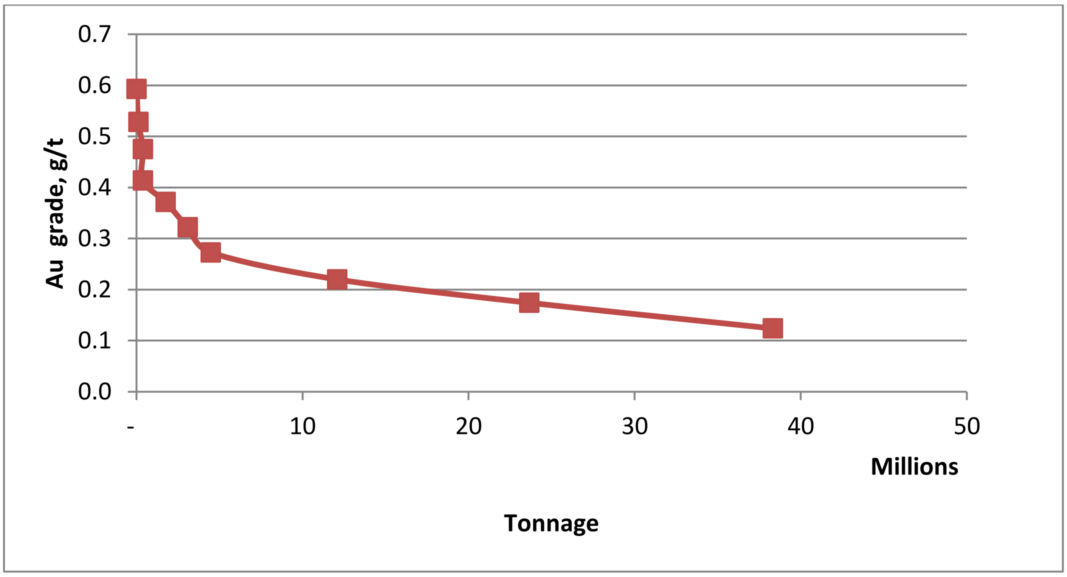

Figure 17 Au grade (%) versus ore tonnage, Mt (ordinary kriging, zoning applied).

The same analysis has been conducted for the ore body model obtained with ordinary kriging (both zoned and unzonned) and inverse distance methods for the ore body within 0,2, 0,3 and 0,4 cut-off grade solid (Table 17, Table 18).

Table 17 Cu grade versus tonnage analysis for 0,2%, 0,3%, 0,4% Cu cut-off.

Within Cut-off solid	Method	Tonnage, Mt	Grade, %	Cu content, Mt
0,2	Ordinary kriging, zonned	342,06	0,413	1,41
0,2	Ordinary kriging, unzonned	342,04	0,411	1,41
0,2	Inverse distance	343,02	0,411	1,41
0,3	Ordinary kriging, zonned	240,01	0,481	1,15
0,3	Ordinary kriging, unzonned	240,59	0,479	1,15
0,3	Inverse distance	241,59	0,478	1,16
0,4	Ordinary kriging, zonned	150,40	0,561	0,84
0,4	Ordinary kriging, unzonned	148,78	0,559	0,83
0,4	Inverse distance	149,31	0,557	0,83

Table 18 Au grade versus tonnage analysis for 0,01 g/t Au cut-off.

Method	Tonnage, Mt	Grade, g/t	Au content, t
Ordinary kriging, zonned	84,27	0,176	14 796
Ordinary kriging, unzonned	84,77	0,176	14 883
Inverse distance	84,21	0,177	14 924

4.6.4. Net smelter return estimation

Net smelter return (NSR) is defined as the proceeds from the sale of mineral products after deducting all off-mine costs related to the transportation, treatment and sale of those products (Goldie R., 1991):

$$NSR = \left(\sum_{i=Au,Cu} \left(G_{i,Conc} - UD_i \right) \cdot P_i \cdot PAY_i - \left(TC + C_{sel} + C_{tran} \right) \right) \cdot \frac{Re_{Cu}}{G_{CuConc}} \qquad (7)$$

In this case following parameters were used for NSR calculation:

- P_{Cu} - Cu price – 6 173 $/t. Price was defined as the average for the period of 2014-22 adjusted for the inflation (World Bank, 2014);
- P_{Au} - Au price – 1 160 $/oz Price was defined as the average for the period of 2014-17 adjusted for the inflation (World Bank, 2014);
- Re_{Cu} - Cu recovery – 85%, which is the value required by the agreement between

Outotec and RCC;

- $G_{Cu,Conc}$ - Cu grade in concentrate – 21.8%, which is the value required by the agreement between Outotec and RCC;
- UD_{Cu} – Unit deduction Cu – 1%;
- UD_{Au} – Unit deduction Au – 1 ppm;
- PAY_{Au} – Au payable 95%;
- PAY_{Cu} – Cu payable 100%;
- TC – treatment charge – 80$/t. This charge was chosen as arithmetic mean between four (4) values: the current treatment charge of 100 $/mt (REUTERS, 2014), the average annual treatment charge for the year 2013 – 70$/mt (Hur, 2013), 2012 - 62 $/mt and 2011 – 85 $/mt (Rojas D.S. et al, 2012):

$$TC = \frac{\sum_{i=1}^{n=4} TC_i}{n} = \frac{\left(100\left[\frac{\$}{t}\right] + 70\left[\frac{\$}{t}\right] + 62\left[\frac{\$}{t}\right] + 85\left[\frac{\$}{t}\right]\right)}{4} \approx 80 \left[\frac{\$}{t}\right] \tag{8}$$

where:

C_{sel} – selling cost – 6 $/t. Selling cost is approximately 10% of the metal's cost in the ore, (as it was done in the (Euromax resources, 2013)):

$$C_{sel} = \frac{P_i}{100}\% \cdot 10\% \approx 6\ [\$/t] \tag{9}$$

- C_{tran} – transportation – 53,66 $/t. Transportation cost is taken directly from the (Alferov et al, 2010) and adjusted for the inflation;
- $G_{cut\text{-}off}$ – cut-off grade – 0,2%, 0,3%, 0,4%.

Where for different cut-off grades, NSRs are following:

$$\begin{cases} G_{cut-off} = 1\% \rightarrow NSR_{1.0\%} = 48{,}80\ [\$/t] \\ G_{cut-off} = 0{,}2\% \rightarrow NSR_{0.2\%} = 9{,}76\ [\$/t] \\ G_{cut-off} = 0{,}3\% \rightarrow NSR_{0.3\%} = 14{,}64\ [\$/t] \\ G_{cut-off} = 0{,}4\% \rightarrow NSR_{0.4\%} = 19{,}52\ [\$/t] \end{cases} \tag{10}$$

4.6.5. Hardness estimation

The only available hardness data for the current ore body was drilling rate that ranged from extremely low to extremely high. Represented as hard, very hard and extremely hard (high, very high and extremely high drilling rate) this data comprised more than 90% of the ore body excluding oxidized ore (Figure 18). The amounts of ore of different hardnesses, which is shown in the Figure 18, were calculated from the block model and based on the geological data base information.

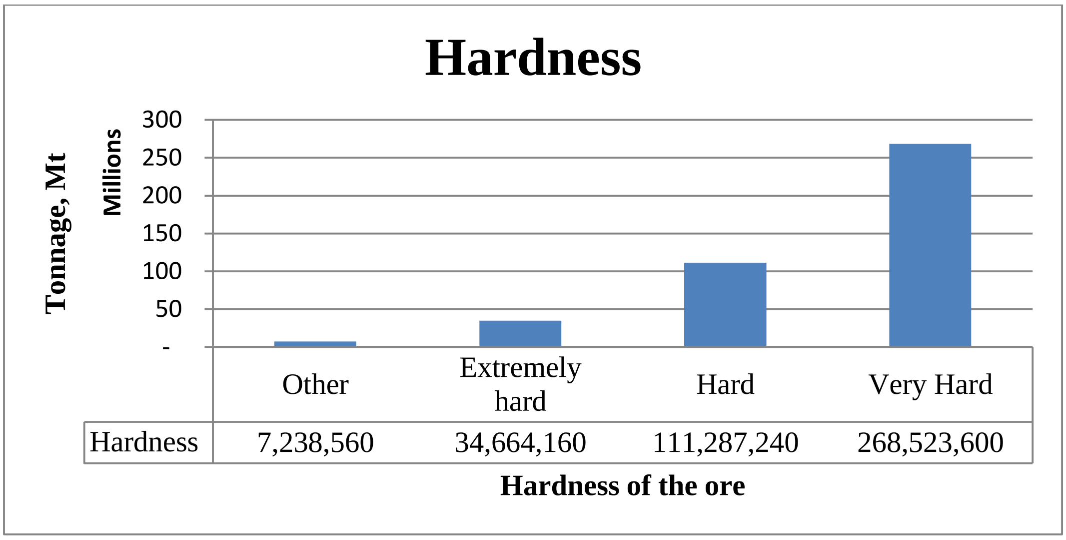

Figure 18 Hardness of the ore versus tonnage (Mt).

Drilling rate, however, is not common in Russia. Instead, Protodyakonov's classification (Heiniö, 1999) is more common for measuring the drillability and blastability. Although drilling rate was used as hardness rate in this Master thesis, it would be recommended to use Bond index in the future investigations. Bond index is more difficult and costly to measure but it is more applicable in further calculations and experiments applied in the mineral processing, mainly in milling.

4.7. Geometallurgical modeling and zoning

In order to improve feed forecasting quality, the ore was classified into 13 types (listed in the Table 19) depending on hardness, iron content (magnetite content), oxidation and presence of Cu content above cut grade of 0,2% . Cu content was not one of the main parameters, since appropriate Cu content in the feed depends on the separate agreements between open pit and the cocentrator plant.

There were three types of hardness: high hardness, very high and extremely high; two types of iron content: high probability of iron content and low probability of iron content; two types of oxidation: primary ore and mixed ore; two types of "Cut-off 0,2% Cu": block is located inside the cut-off solid and outside the cut-off solid.

Table 19 Ore types.

Cut-off 0,2% Cu	Hardness	Magnetite	Oxidation	Code
Outside	Not relevant			1
Inside	Hard			2
Inside	Hard		X	3
Inside	Hard	X		4
Inside	Hard	X	X	5
Inside	Very hard			6
Inside	Very hard		X	7
Inside	Very hard	X		8
Inside	Very hard	X	X	9
Inside	Extremely hard			10
Inside	Extremely hard		X	11
Inside	Extremely hard	X		12
Inside	Extremely hard	X	X	13

Geometallurgical ore type with the code 13 remained only theoretical type of ore and was not observed in the ore body.

4.8. Open pit optimization

While the usual economical open pit optimization involves only variation of the mining expence with the depth, the geometallurgical approach introduces changing ore processing cost (according to ore type) as the second variable. Such more detailed approach allows to have more realistic approach to the concentrator revenues during it lifetime.

Four different open pit optimization scenarios were modelled:

1. Open pit with cut-off grade 0,2% Cu (NSR = 09,76 $/t)
2. Open pit with cut-off grade 0,3% Cu (NSR = 14,64 $/t)
3. Open pit with cut-off grade 0,4% Cu (NSR = 19,52$/t)
4. Open pit with cut-off grade 0,2% Cu (NSR = 09,76 $/t) applied only to the Northern and Central parts of the ore body.

Cut-off (NSR) was the only unique parameter which was different for Scenarios 1-3. In case of Scenario 4 calculations were also limited to the Northern and Central zones of the ore body, since Northern and Central zones of the ore body have much higher metal content than Sothern zone has.

A set of parameters was used to perform the open-pit optimization:

- Twelve ore types were used as main ore types for the open pit optimization;
- Topography was using real topography model before the excavation started; the file's name is topo_may2006.dtm;
- The base level was chosen to be -150 m;
- Optimizations was made for the block size of 40 × 40 m;
- Optimization was done for discounts, (%, represents stage in the open pit development in per cents from the ultimate pit), every 2%: 0, 2, 4 … 92;
- The default slope angle was 48°;
- Sale recovery 99%;
- Milling recoveries and costs as per (Table 20);
- Mining cost changes with elevation as per (Table 21).

Table 20 Milling recoveries and costs

Material	**Milling Recovery[2], %**	**Milling Cost, $/t**	**Material**	**Milling Recovery[2], %**	**Milling Cost, $/t**
1	0	8,621	**7**	80	8,621
2	100	10,561	**8**	83	10,561
3	90	8,621	**9**	73	8,621
4	93	10,561	**10**	85	10,561
5	83	8,621	**11**	75	8,621
6	90	10,561	**12**	78	10,561

Table 21 Mining cost changes with elevation

[2] due to specifics of the optimization software, milling recoveries correspond to the deduction in milling cost, but not to the recoveries by the technological process. Recovery of 85% is already assumed in the milling costs.

Elevation, m	Cost, $/m^3	Elevation, m	Cost, $/m
270	3,00	**30**	4,14
210	3,28	**-30**	4,42
150	3,57	**-90**	4,70
90	3,85		

However, for this study the most important optimized pits were those, which corresponded to the first 5,5 years of the ore extraction (Figure 19). Open pit optimization resulted in a number of pit shells with different geometallurgical ore type distributions which are illustrated in the Figure 19 and results for the one of them is presented in Table 22.

Table 22 Open pit optimization results (0,2% cut off grade).

Material Name	Volume ×10^3, m^3	Tonnage, Mt	Cu, %	Au, g/t
1	107 524	295,83	0,038	0,006
2	23 832	66,97	0,355	0,052
3	332	0,73	0,400	0,071
4	11 968	33,63	0,442	0,077
5	68	0,15	0,513	0,045
6	53 892	151,44	0,357	0,049
7	284	0,63	0,385	0,068
8	28 932	81,30	0,402	0,059
9	76	0,17	0,546	0,078
10	7 816	21,96	0,377	0,062
11	24	0,05	0,397	0,068
12	2 388	6,71	0,456	0,041
Total, rock	**237 136**	**659,56**	**0,225**	**0,033**
Total, ore	**129 612**	**363,74**	**0,378**	**0,055**

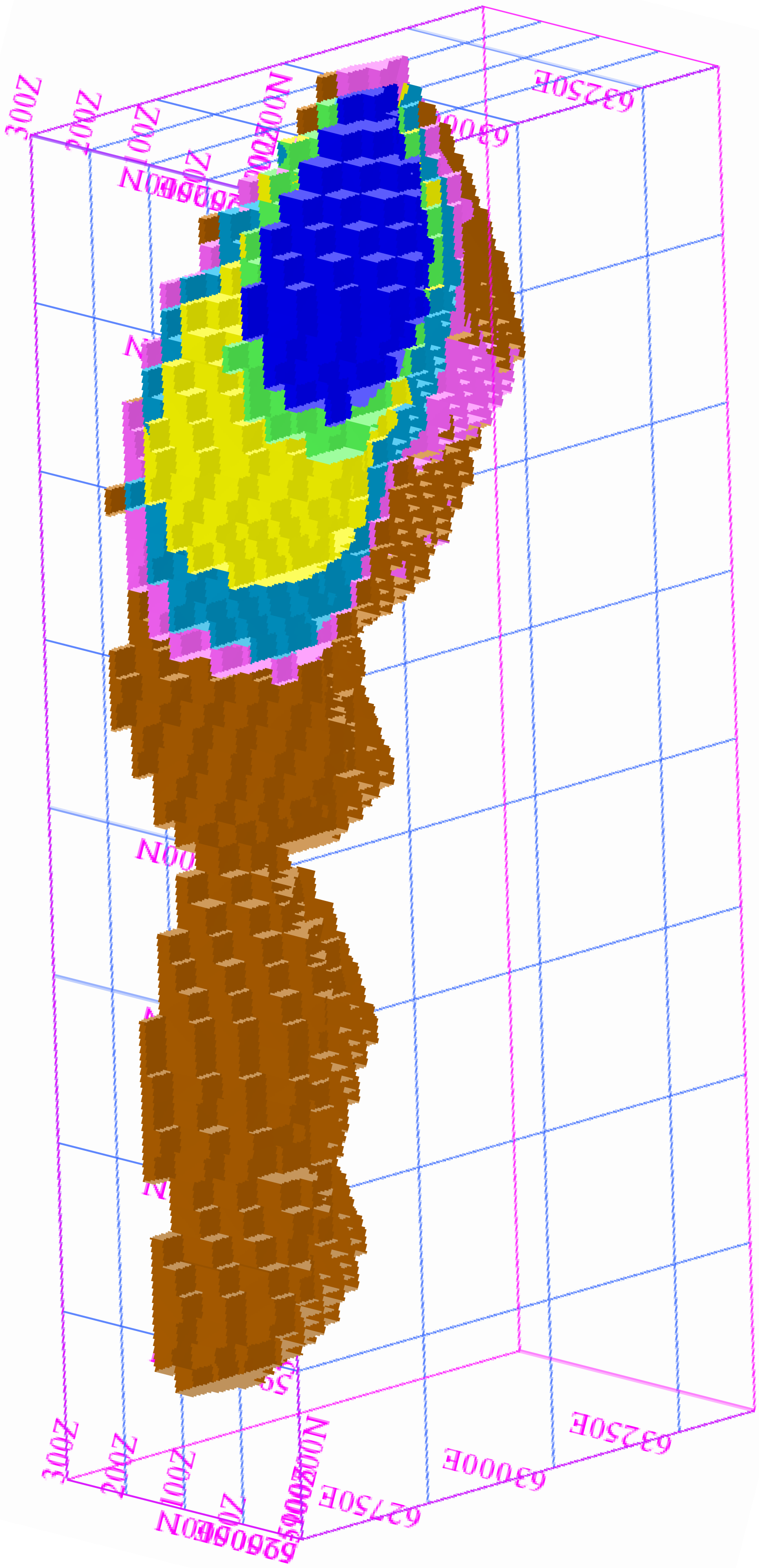

Figure 19 Open pit optimization: open pit progress, stages 1-5 and stage 9 (Surpac 6.3).

4.9. Open pit design

Five open pit designs were created for the following cases. (Thus only three are presented below in the Figure 20):

1. The ultimate open pit based on the scenario 1 (Cu cut-off grade 0,.2%);
2. The ultimate open pit based on the scenario 2 (Cu cut-off grade 0,3%);
3. The ultimate open pit based on the scenario 3 (Cu cut-off grade 0,4%,);
4. The open pit based on the scenario 1 (Cu cut-off grade 0,2%) for the year 9 (up to year 10);
5. The open pit based on the scenario 1 (Cu cut-off grade 0,2%) for the year 5 (up to year 6);

Open pit design resulted in geometallurgical ore type distribution for four different scenarios (results for two of them are given in Table 23, Table 24) corrected for the smoothening and ramp presence. Designed open pits (Figure 20) were, on average, 20% larger than optimized (modeled) open pits due to smoothening and ramp addition. Cu and Au content was relatively 1-1,5% lower in designed open pits than in optimized models, which was also due to smoothening and ramp addition.

Table 23 Open pit design results (0,2% cut off grade).

Material Name	Volume $\times 10^3$, m^3	Tonnage, Mt	Cu, %	Au, g/t
1	163 464	452,20	0,025	0,004
2	23 964	67,34	0,352	0,051
3	340	0,75	0,396	0,069
4	12 012	33,75	0,442	0,077
5	68	0,15	0,513	0,045
6	54 912	154,30	0,352	0,048
7	284	0,63	0,385	0,068
8	29 444	82,74	0,400	0,058
9	76	0,17	0,546	0,078
10	7 968	22,39	0,374	0,061
11	24	0,05	0,397	0,068
12	2 448	6,88	0,450	0,040
Total, rock	**295 004**	**821,35**	**0,1820**	**0,0266**
Total, ore	**131 540**	**369,15**	**0,3745**	**0,0545**

Table 24 Open pit design results (0,2% cut off grade – 5 years).

Material Name	Volume ×10^3, m^3	Tonnage, Mt	Cu, %	Au, g/t
1	41 032	112,63	0,036	0,006
2	6 292	17,68	0,392	0,049
3	680	0,15	0,439	0,076
4	4 280	12,03	0,511	0,078
5	28	0,06	0,499	0,074
6	16 284	45,76	0,405	0,051
7	152	0,34	0,422	0,062
8	13 036	36,63	0,463	0,068
9	36	0,08	0,641	0,129
10	972	2,73	0,577	0,084
11	8	0,02	0,517	0,051
12	1 088	3,06	0,572	0,041
Total, rock	**83 276**	**231,16**	**0,243**	**0,033**
Total, ore	**42 244**	**118,53**	**0,440**	**0,059**

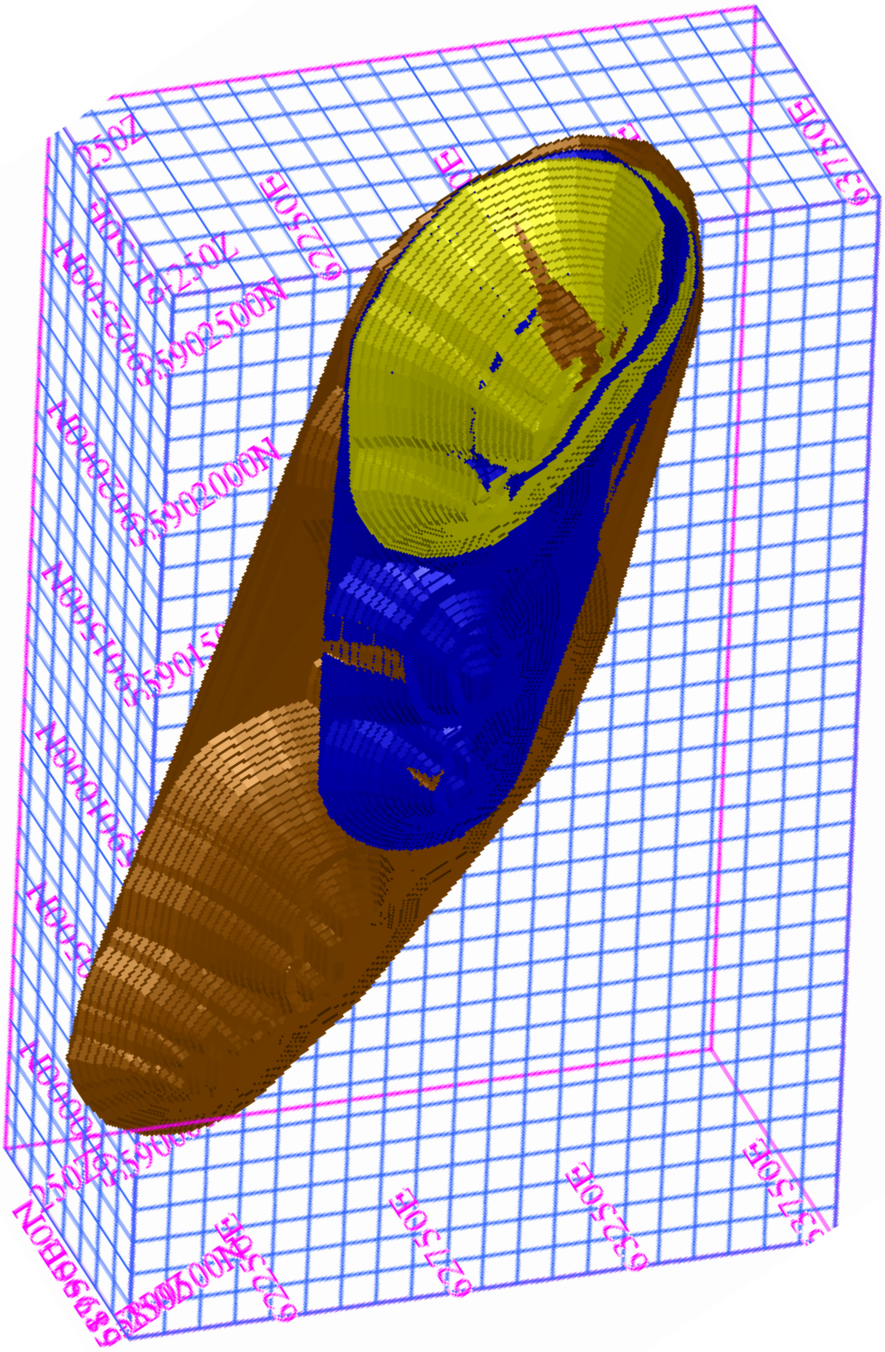

Figure 20 Open pit design: open pit cut-off 0,2% Cu (ultimate open pit, open pit after 9 years of exploitation and open pit after 5 years of exploitation). (Surpac 6.3)

5. Cost estimation and financial analysis

5.1. Cost estimate

The purpose of cost estimation was to define the profitability of the project. The methodology of the cost estimation was based on the cash flow planning. One annual quarter was chosen as the time step for the analysis. The calculations were run for 5,5 years. Year 0 was the year when construction was finished. Year 1 was the first year of commercial exploitation of the deposit.

Extraction tax was 8%, amortization – 0,09 $/t, general production costs – 0,35$/t, VAT,% - 0%. The total capital cost of the processing plant and open pit construction was equal to 0,8 b$. Discount rate was adopted at the level of 2,5 % per quarter.

Milling costs (Table 25) depended on the geometallurgical ore type. It was assumed that hardness “Hard” did not change milling costs, while “Very Hard” increased costs by 10% and “Extremely Hard” by 15%. Assumption was based on the consultations with Outotec site engineers at Mikheevskoye site and have to be verified during the actual production. Magnetite presence increased milling costs by 7,5% and oxide presence (transitional ore presence) increased milling cost by 10% (values are based on the discussions with the Outotec development metallurgist at the Mikheevskoye site). The basic (minimum) milling cost for the primary ore was 8,62 $/t and for the transitional ore – 10,56 $/t. Values are based on the costs proposed by Outotec engineers from the Mikheevskoye site. The average cost for the primary ore was 9,00 $/t and for the transitional ore – 11,76 $/t.

Table 25 Milling costs.

Ore type	Milling cost, $/t	Ore type	Milling cost, $/t
1	N/A	**7**	12,67
2	8,62	**8**	10,13
3	11,62	**9**	13,47
4	9,27	**10**	9,91
5	12,41	**11**	13,20
6	9,48	**12**	10,56

Mining costs (Table 26) depended on the depth H and were calculated as follows:

$$Mining\ cost\ [\$/m^3] = -0{,}00474 \cdot H + 4{,}27720 \tag{11}$$

Table 26 Mining cost changes with elevation.

Elevation, m	Cost, $/m³	Elevation, m	Cost, $/m³
270	3,00	**30**	4,14
210	3,28	**-30**	4,42
150	3,57	**-90**	4,70
90	3,85	**-**	-

5.2. Mine scheduling

Two methods were used for the mine scheduling. The first method was based on the geometallurgical ore type. In this method the ore was excavated gradually by the geometallurgical ore types. The second method was based on Cu content: ore was excavated gradually by the Cu content. In both scenarios, oxides had to be extracted first and primary ore had to be excavated afterwards. Copper concentrate with Cu grade of 21,8% was adopted as the final product. Recovery for Cu was 85% and for Au - 65%.

5.2.1. Case scenario 1

Schedule for the case scenario 1 (Appendix 17, Table A17 - 1) was based on application of geometallurgical zoning in mine planning. Schedule was prepared by annual quarters. Extraction was planned for five and half years and in five stages. Only central and Northern parts of the ore body were considered for mining and extraction. The total weight of the extracted rock was estimated to be 159,40 Mt with average 0,453 % Cu and 0,06 g/t Au. The total amount of waste rock was 65,69 Mt, ore – 93,24 Mt, transitional ore – 0,47 Mt. Mine schedule revealed that both Cu and Au content would tend to decrease during the first 6 years of mine extraction.

Next, Cu grade plan (Appendix 17, Table A17 - 4), Cu tonnage plan (Appendix 17, Table A17 - 3), Au grade plan (Appendix 17, Table A17 - 4) and Au tonnage plan

(Appendix 17, Table A17 - 5) were developed on the basis of the mining schedule (Appendix 17,Table A17 - 1). NSR ((7), page 53) and mine schedule (Appendix 17,Table A17 - 1) were used to calculate revenues (Appendix 17, Table A17 - 6).

Milling costs from the (Table 25), mining costs from the (Table 26) and mining schedule (Appendix 17,Table A17 - 1) were used to define the milling cost plan (Appendix 17, Table A17 - 7) and mining cost plan (Appendix 17, Table A17 - 8). Financial analysis is presented in (Appendix 17, Table A17 - 9) and cash flows are shown graphically in Figure 21.

Cash flow, discounted cash flow, accumulated cash flow and accumulated discounted cash shown in Figure 21 demonstrate that it will take more than five years for the project to pay back under the current market situation and above described conditions.

It will take more than five years for the project to pay back under the current market situation and above described conditions.

5.2.2. Case scenario 2

Schedule for the case scenario 2 (Appendix 18, Table A18 - 1) was based on extracting higher Cu content ore first. Schedule was prepared by quarter. Extraction was planned for five and half years and in five stages. Only central and northern parts of the ore body were considered. The total weight of the extracted rock was estimated to be 159,40 Mt with average 0,45 % of Cu and 0,06 g/t of Au. The total amount of waste rock was 65,69 Mt, ore - 93,24 Mt, transitional ore – 0,47 Mt (the same as in the case scenario 2).

Mining schedule revealed that both Cu and Au content would be decreasing during the first 6 years of operation.

Next, Cu grade plan (Appendix 18, Table A18 - 2), Cu tonnage plan (Appendix 18, Table A18 - 3), Au grade plan (Appendix 18, Table A18 - 4) and Au tonnage plane (Appendix 18, Table A18 - 5) were developed on the basis of the mining schedule (Appendix 18, Table A18 - 1). NSR ((7), page 53) and mining schedule (Appendix 18, Table A18 - 1) were used to calculate revenues (Appendix 18, Table A18 - 6).

Milling costs from (Table 25), mining costs from (Table 26) and mining schedule (Appendix 18, Table A18 - 1) were used to calculate the milling cost plan (Appendix 18, Table A18 - 7) and mining cost plan (Appendix 18, Table A18 - 8). Detailed financial analysis is presented in (Appendix 18, Table A18 - 9) and cash flows are graphically shown in Figure 21.

It will take more than five years for the project to pay back under the current market situation and above described conditions.

5.2.3. Comparison of the scenarios

Although case scenario one and case scenario two are very similar and look very similar in
Figure *21*, some small differences still can be observed between proposed scenarios. The Figure *21* compares cash flows (CF), discounted cash flows (DCF), net present value (NPV) and accumulated cash (Accum) flow for both scenarios for 5,5 years (22 quarters). However, capital costs for the year zero are not shown in the
Figure *21*.

Both case scenarios show similar trends. Cash flow and discounted cash flow are expected to increase during the first five quarters and reach its peak in the quarter five. The largest decline is expected between quarters six and ten. There is a plateau in quarters 10-22 and no big fluctuations are expected in following periods.

NPV and accumulated cash flow for both scenarios increase quite quickly from quarter 1 to quarter 10. Period from quarter 11 to 22 show slightly slower increase.

The case scenario one shows slightly better performance than case scenario two in terms of cash flow and NPV for the whole period of 22 quarters.

5.2.4. Schedule plan discussion and key findings

Mine schedules were designed in the way that amount of waste (“waste”), primary ore (“ore”), and transitional ore (“oxides”) were the same for the both mining plans. Material extraction was planned to be increasing during the 5,5 initial years of operation.

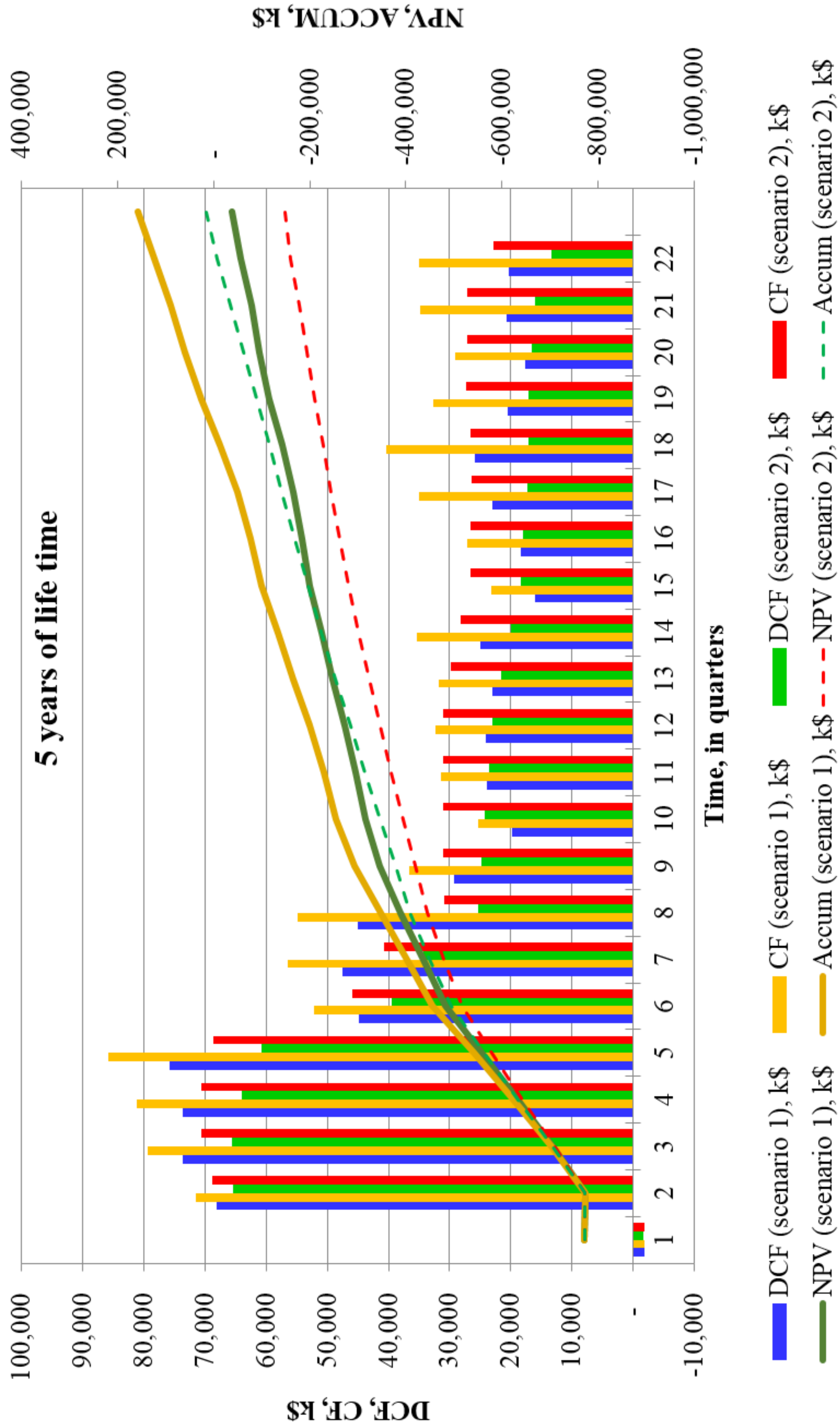

Figure 21 Cash flows, discount cash flows, Net present value and Accumulated NPV in k$, comparison of the case scenarios 1 and case scenario 2 .

Cu grade changes during the initial 5,5 years of operation were assumed to be mainly within the range of 0,3-0,6 % Cu (red lines in Figure 22) with the average of 0,45% Cu (green line in Figure 22). Cu grade change is shown in Figure 22 with a brown line for the case scenario 1 and the blue line in Figure 22 reflects Cu grade for the case scenario 2. Cu grade difference between two case scenarios may be due to low time resolution of the schedules, however, Cu grade curves for both scenarios are quite close to each other. So, extraction and planning of concentrator plant feed based on the geometallurgical ore type zonality is possible.

Concentrate and thus Cu production (Figure 23) depend on the Cu grade (Figure 22) and feed amount.

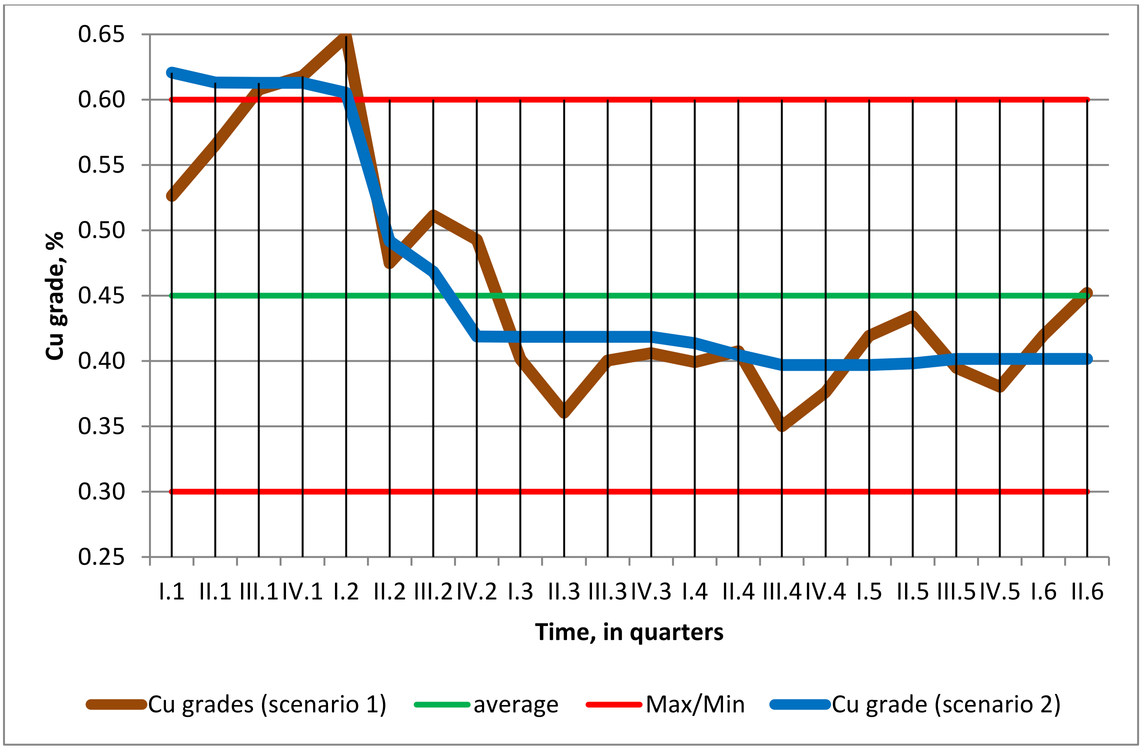

Figure 22 Cu grade forecast according to case scenarios 1 and case scenario2.

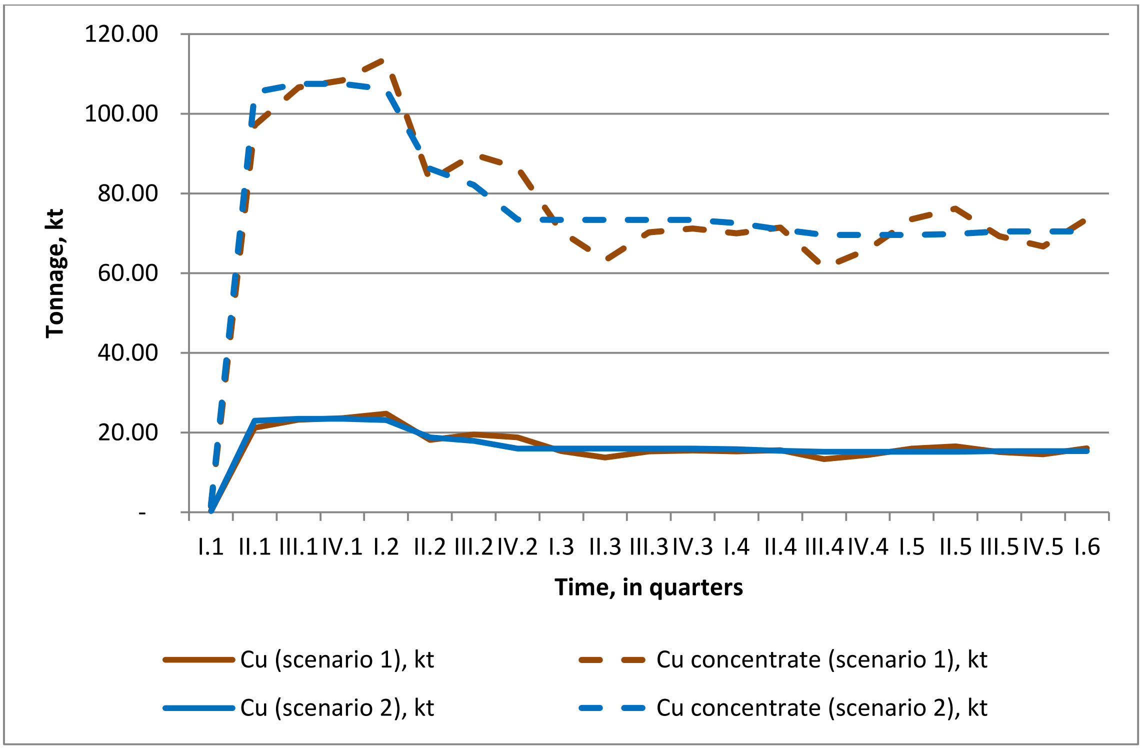

Figure 23 Concentrate and copper production capacities according to scenarios 1 and 2.

Au grade change during the initial 5,5 years of operation is declining (Figure 24). Brown line in Figure 24 describes Au grade change for the case scenario 1 and blue line in Figure 24 describes Au grade change for the case scenario 2.

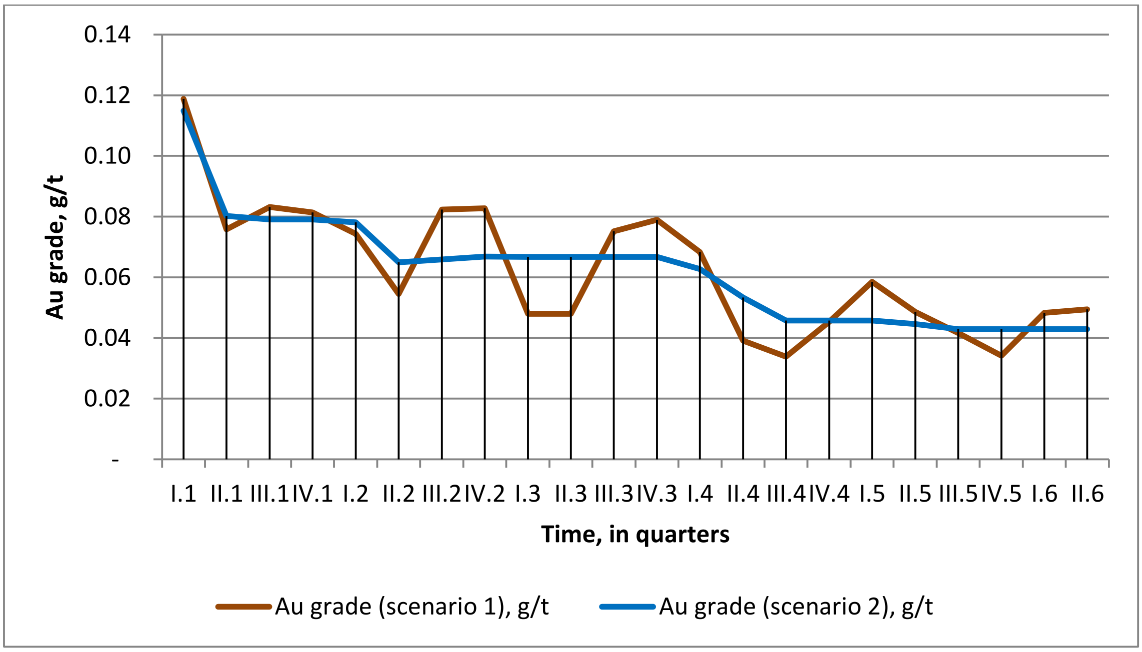

Figure 24 Au grade forecast according to scenarios 1 and 2.

Au production (Figure 25) depends on the Au grade (Figure 24) and feed amount.

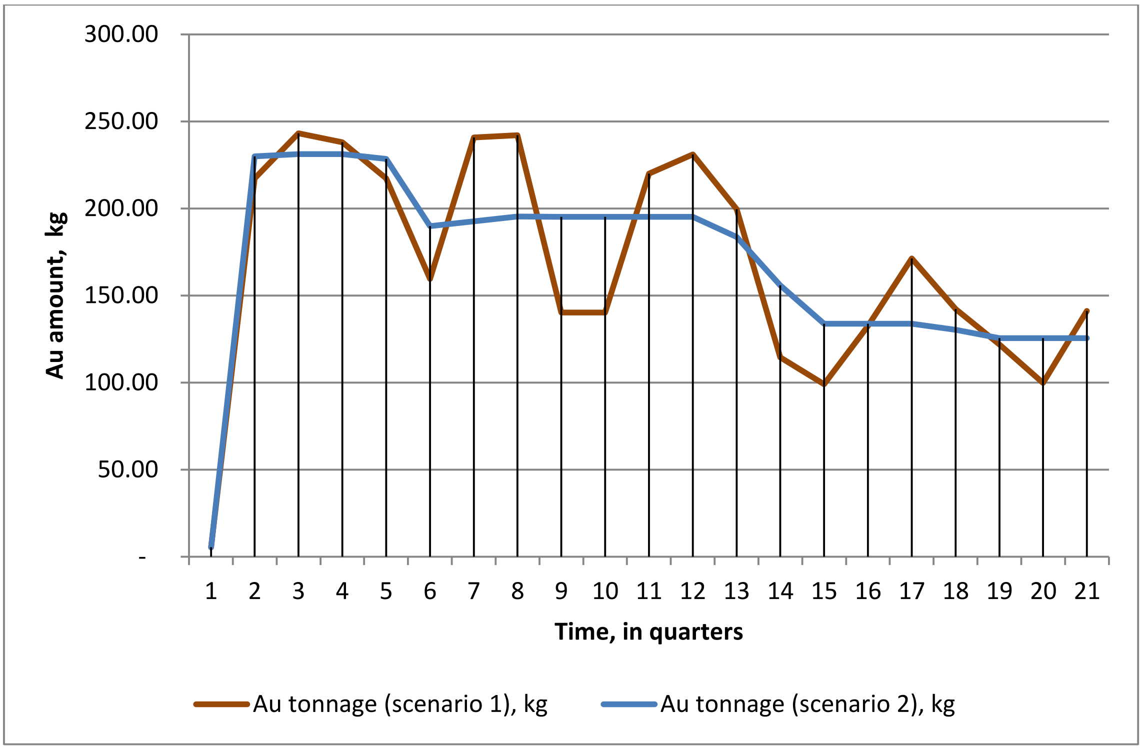

Figure 25 Gold production capacities according to scenarios 1 and 2.

6. Discussion, conclusions and recommendations

6.1. Discussion

Limitations of the work

Several limitations had significant impact on the results of this research. Geological survey network was quite poor and thus geological database was not representative enough. Several mistakes were observed in the database. Some data in the database was not informative at all (many abbreviations could not be decoded).

Hydrothermal alteration zonality could not be properly applied for the mine scheduling due to poor database. Sulphide zonality (which is the closest one to the alteration zonality) was applied instead.

Sampling campaign was not supported enough with necessary equipment (truck, excavator, crusher), which could cause some result distortion.

Full scale production did not start by the end of this research. So, data from the controlling and measuring equipment was not available, laboratory test results for plant feed were not reliable, feed quality and location of its extraction during initial months of production were not clear.

The above mentioned limitation made it difficult to verify results of the research, however it is strongly recommended to apply results of the research in the future mine scheduling and mineral processing planning.

The implications of the work for future research

Current results should be used for the second sampling campaign to prove actual existence of the twelve theoretical ore types. Obtained block model and geological (alteration/sulphidation) zonality could be used for locating geometallurgical ore types within ore body. According to **(**Salehian and Ghaderi, 2010**)**, it is recommended to pay great attention to the porphyry geological zonality, since it may reduce exploration costs and influence the exploration costs reduction.

Since database was logged as normal ore type, it is important to insure future logging in accordance with hydrothermal alteration. So, alteration logging has to be performed first.

Feed grade decrease is expected during the following 5,5 years of operation from the average 0,65% Cu to the average 0,45% Cu. Simultaneously, feed quality and processing cost decreases are possible to be achieved with application of zonality based on alterations (in case if Cu-porphyry ores) and geometallurgical zonality.

When mineral processing will run to its full capacity, the processing performance of the different geometallurgical ore types would need to be verified in the beneficiation process. During the final stages of this research project the two significant process performance differences were noticed in practice: i) process feed with and without presence of mixed/ transitional ore (primary sulphide ore is grey in color, mixed or transitional ore has brown color; both color and performance differences are distinguishable throughout all of the minerals processing stages including milling, flotation, thickening and filtration) and ii) rock hardness (differences present as varying throughput in the milling section and significant - up to 50% - change in power draw of the milling section for the same throughput).

6.2. Recommendations

Mikheevskoye project has a unique status for the Russian mining industry due to its huge production scale and low grades. Hence, any improvement in the geological knowledge would result in improved performance of the concentrate production through better production planning and maintenance planning.

Ore bearing coefficients in stock work of the primary sulphide ore vary between 0,83 – 0,21, which means obligatory ore assaying before extraction. Therefore, it is strongly recommended to repeat sampling campaign in accordance with geometallurgical ore type zonality in cooperation with mine surveyor and geologists, and taking into account the most recent mining schedule by MiGOK. It might be beneficial to use Bond Index instead of hardness parameter for fine-tuning geometallurgical ore type zonality. Laboratory test have to be conducted to specify the processing regime for the different ore types. However, these tests cannot be conducted in Finland due to Russian Legislation, which restricts transportation across border of Russian Federation of the Au containing ore. Therefore, all the necessary tests have to be conducted in Russia and preferably in the

Mikheevskoye or other nearby RCC laboratories, which would significantly reduce the cost of the tests.

Additional geological drilling could be performed in the future. Although a lot of exploration works were planned in the project documentation (Bulatov et al, 2010), none of them were performed fully due to economical constraints experienced by RCC at the time of this study. The strategy and implementation of the geological exploration has to be widely discussed with the RCC geologists and surveyors in order to fill in gaps in our understanding of the geological structure of the ore body. Furthermore, in order to utilize method proposed in this study, the logging has to be performed properly in accordance with the hydrothermal alteration of the ore body. As it was stated earlier, good logging database has to present alterations but not just rock types.

Mining schedule presented in this study and based on geometallurgical ore type zonality has to be verified in cooperation with RCC geologists and surveyors and used for the future production planning. Information contained in the block model developed in the course of this study regarding the new approach to the metal distribution evaluation has to be a central instrument for the discussions between stakeholders about the future feed quality.

The procedure described here for definition of the geometallurgical ore type zonality and feed quality forecast can be used in other similar projects. It would be especially valuable for the projects related to the porphyry Cu deposits, where geological hydrothermal alteration zonality can be applied for the metal distribution zoning.

6.3. Conclusions

This study has proved that mine planning can be improved for the Cu-porphyry deposits. As a result, the forecasting tool for the feed quality was developed and it was based on principles of porphyry copper ore zonality and geometallurgical ore type zoning.

Geological zoning was applied in the ore body modeling for the porphyry Cu deposit. Geological zoning was based on the definition of alteration zones within ore body. Since alteration zonality was not distinguishable enough for the purpose of this study from the available data, zoning based on the sulphidation (mineral content) was applied instead.

Strong correlation between alteration zonality and sulphidation was assumed in doing so. Geological zoning for the ore body modeling had proved to be competitive method in comparison with the classic approach (classic approach presumes ore body modeling based on metal content). In classic approach it was clear that basic statistics were different for different geological zones and thus had to be modeled in a different way.

Geometallurgical zoning was applied for the mine planning and production scheduling. Twelve geometallurgical zones were specified based on the geological knowledge of the ore body and mineral processing requirements. It was shown that mine scheduling based on the geometallurgical zoning is possible and feasible.

In this study, forecasting tool was developed for the Mikheevsky ore deposit, however, it can be applied to most other Cu-porphyry deposits all over the world. Since, alteration in Mikheevskoye deposit went not so deeply as normally porphyry does. Therefore, applying hydrothermal zonality in the process of developing mine schedule might have even greater effect on the cases where hydrothermal alterations are more clear and thus contribute significantly to the improvement of the mill performace.

In addition, one of the key findings was that pay back time for the project will be 7,5 years according to the case scenario one and more than 9 years according to the case scenario 2 under the current market situation and above described conditions. Pay back time estimated in this thesis is longer than initially declared minimum pay back time of five years (IMC Economic and Energy Consulting Limited, 2008). However, it is shorter than expected period of 13 years declared in (Alferov et al, 2010).

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Table of figures

Table of tables

Table of appendices

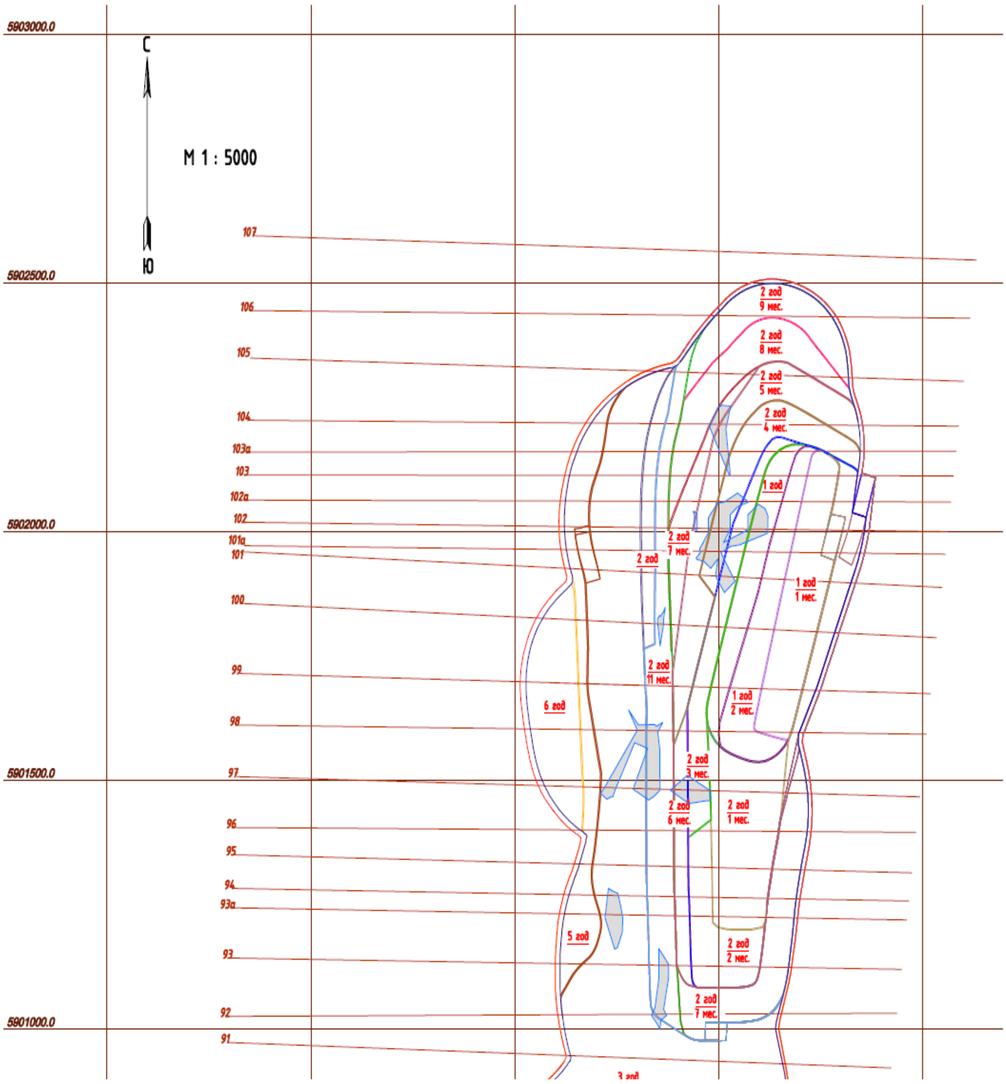
Appendices

Appendix 1 Geological exploration drilling

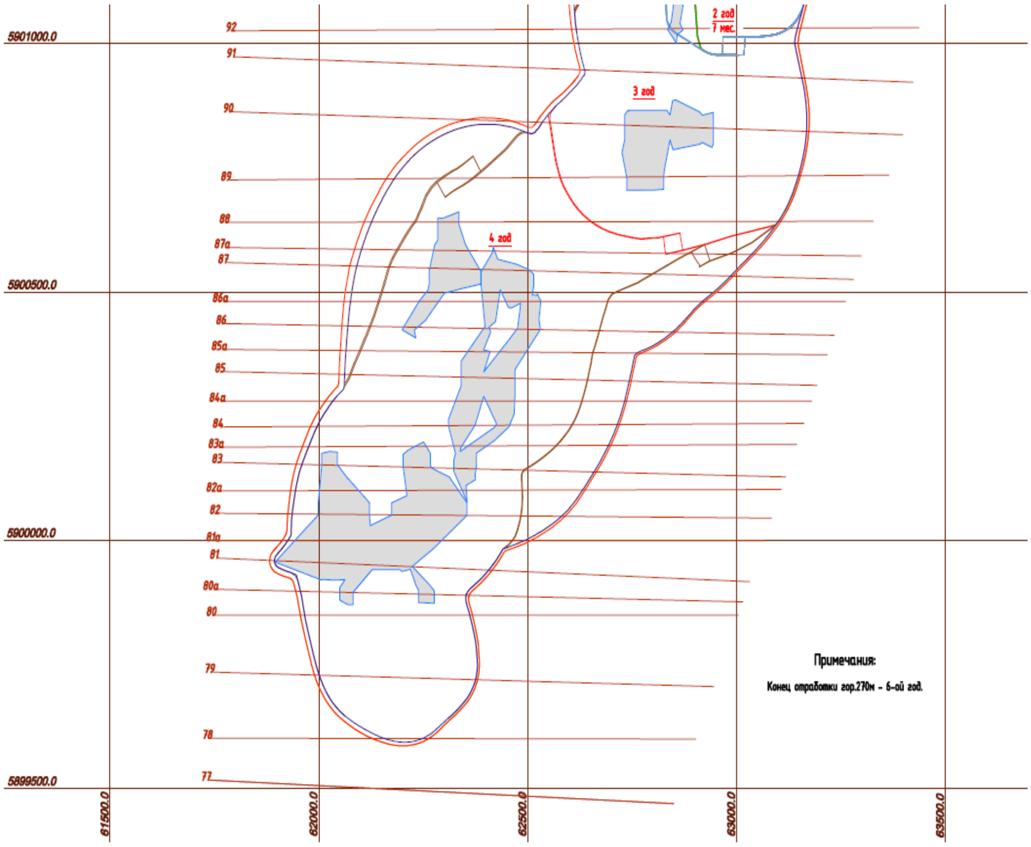
Year	Number of holes	Area	Drilling type	Depth	Network	Purpose
1959	23	South Miheevskoye	Rotary hydraulic	20-30 m	1 km x 200 m	Geochemical Survey
1961	6	South Miheevskoye	Diamond	250-300 m	Targeted	Exploration
1966	9	Central Miheevskoye	Diamond	250-300 0m	Targeted	Exploration
1982	42	All Miheevskoye	Diamond	250-300 m	200 x 200 m	Exploration
1984	80	All Miheevskoye	Rotary hydraulic	20-40 m	100 x 200 m	Geochemical Survey
1986	46	North Miheevskoye	Diamond	150-300 m	200 x 200 m	Resource definition
1994	6	Profile 103	Diamond	300 m	1 profile	Resource Definition
1997	3	Profile 102	Diamond	300 m	1 profile	Resource Definition
1998	22	North Miheevskoye	Diamond	150-300 m	200 x 100 m	Resource Definition
1999	30	Central and South Miheevskoye	Diamond	150-300 m	200 x 200 m	Resource Definition
2000	56	All Miheevskoye	Diamond	150-300 m	100 x 100 m	Resource Definition
2005	5	All Miheevskoye	Diamond NQ	150-220 m	Targeted	Confirmation Drilling
2005	7	All Miheevskoye	Diamond PQ	100-150 m	Targeted	Metallurgical PFS
2005	34	All Miheevskoye	Diamond NQ	100-500 m	100 x 50 m North and 100 x 100 m south of 5901810N	JORC Inferred Resource
2006	12	All Miheevskoye	Diamond NQ	100-230 m	targeted	Metallurgical DFS
2006	40	All Miheevskoye	Diamond NQ	100-400 m	100 x 50 m North of5901810N100 x 100 m South of 5901810N &North of 5900630Nand100 X 50 m South of 5900630N	JORC Inferred Resource
2006	8	All Miheevskoye	Diamond NQ	150-350 m	50 x 50 m North Zone	Infill section JORC Measured Resource
Total	429					

Appendix 2 Mining plan. Northern block.

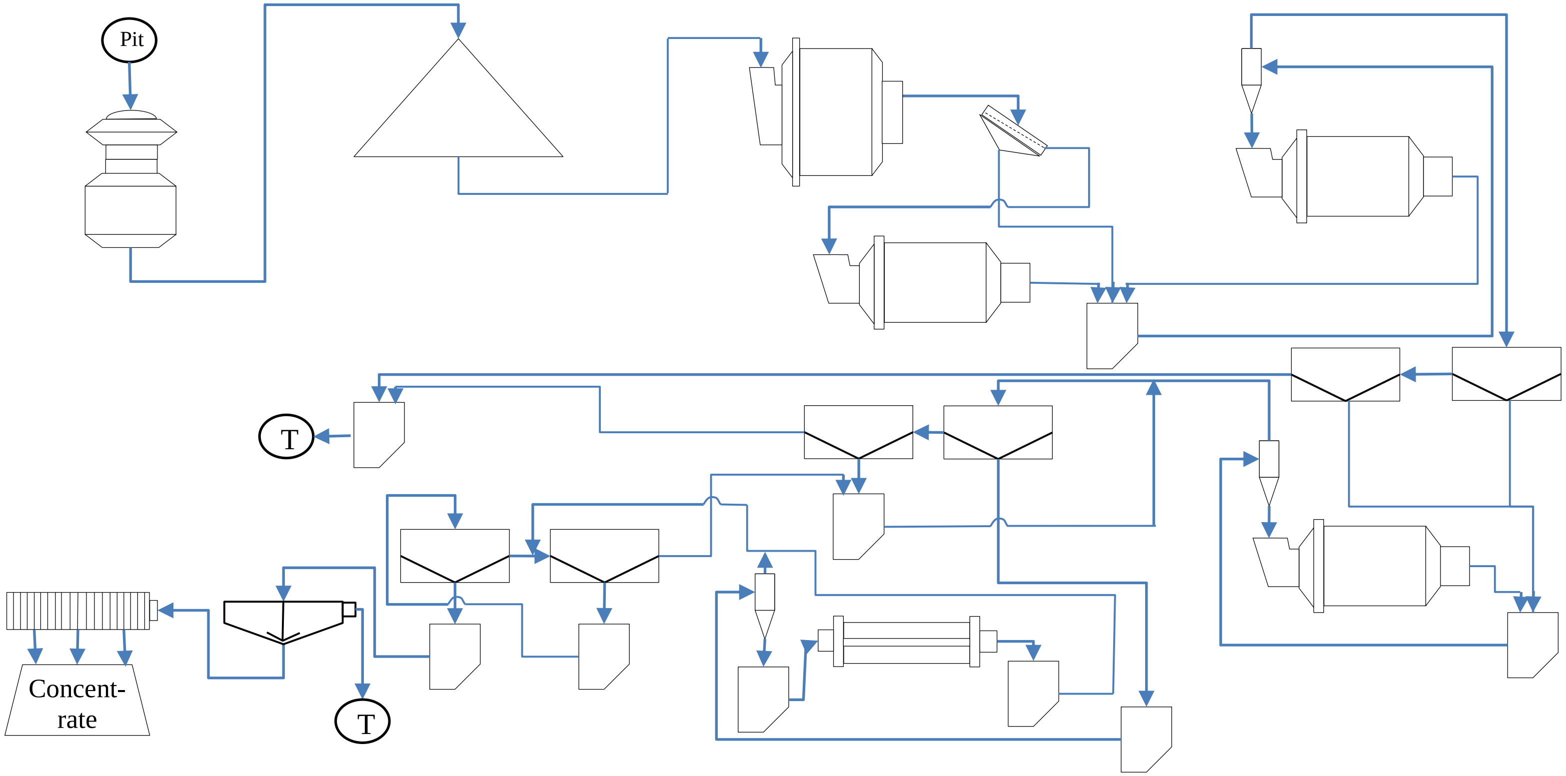
План отработки горизонта 270 м.

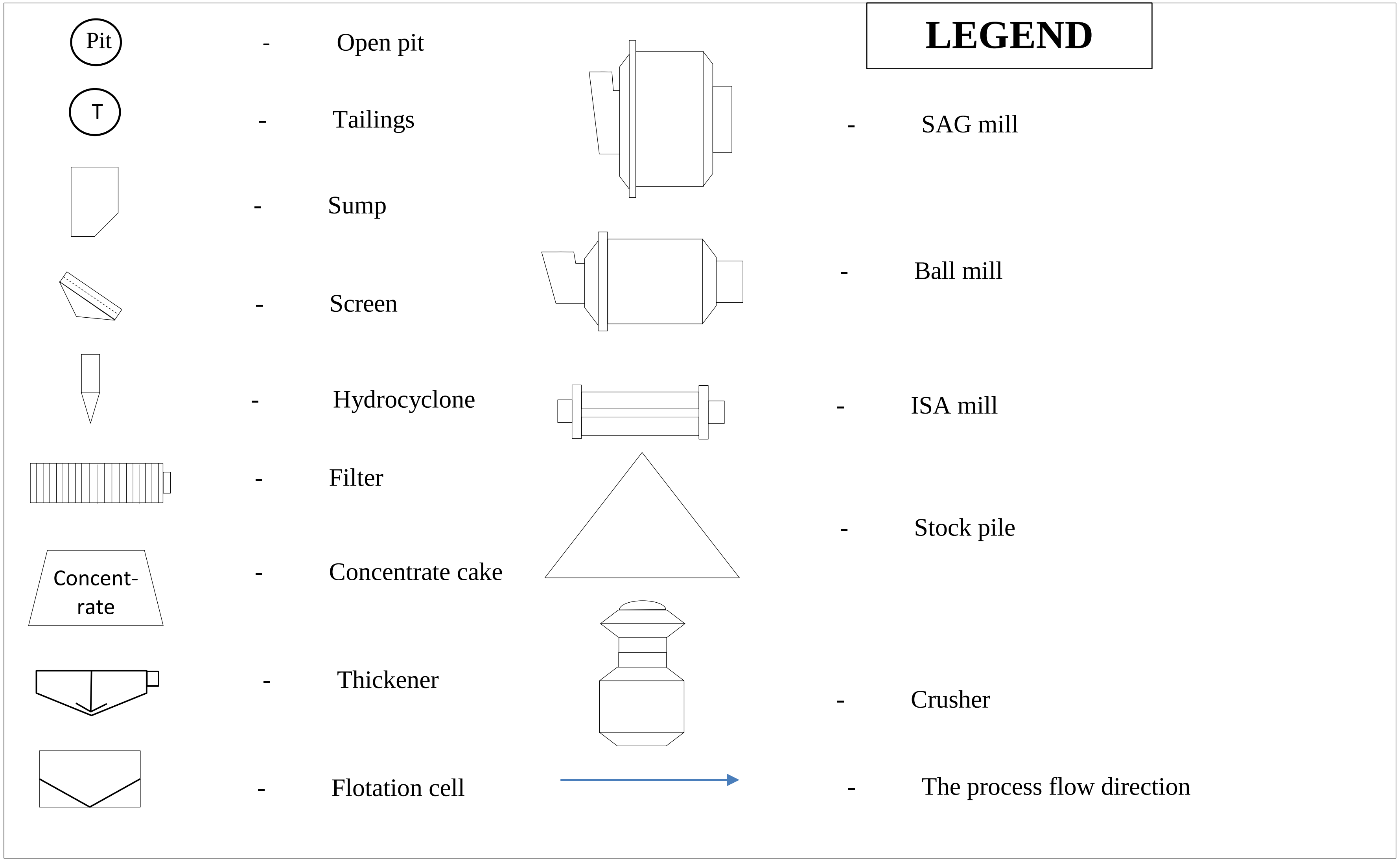


Appendix 3 Mining plan. Southern and Central block.



ЗАО "Михеевский ГОК"
Планы горных работ


Pit
T
T
Concent-
rate


LEGEND
Pit
- Open pit
T
- Tailings
- Sump
- Screen
- Hydrocyclone
- Filter
Concent-
rate
- Concentrate cake
- Thickener
- Flotation cell
- SAG mill
- Ball mill
- ISA mill
- Stock pile
- Crusher
- The process flow direction

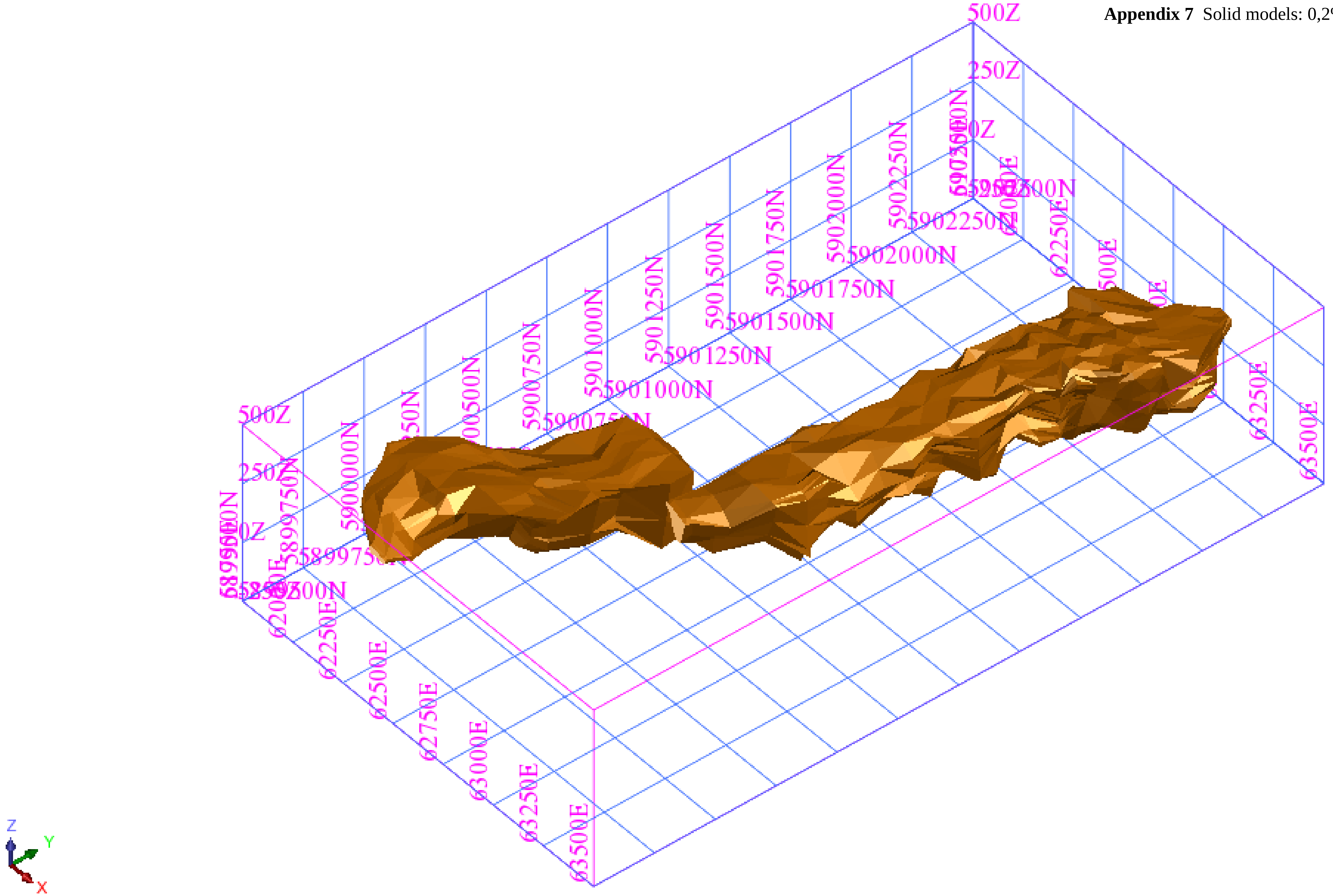
Appendix 5 Set_2010_Comb database extensions.

	Hole Id	**Northing**	**Easting**	**Elevation**	**Depth**
Min. Northing	2269	5898790	62600	272,40	151,1
Max. Northing	MG1	5907566	65393	277,00	7,0
Min. Easting	S006	5900500	61140	280,00	100,0
Max. Easting	MG20	5902420	69516	277,00	5,0
Min. Elevation	714	5899624	62846	272,07	19,2
Max. Elevation	843	5900437	61920	284,30	31,5
Min. Depth	MG20	5902420	69516	277,00	5,0
Max. Depth	M011	5902010	63100	282,90	500,1

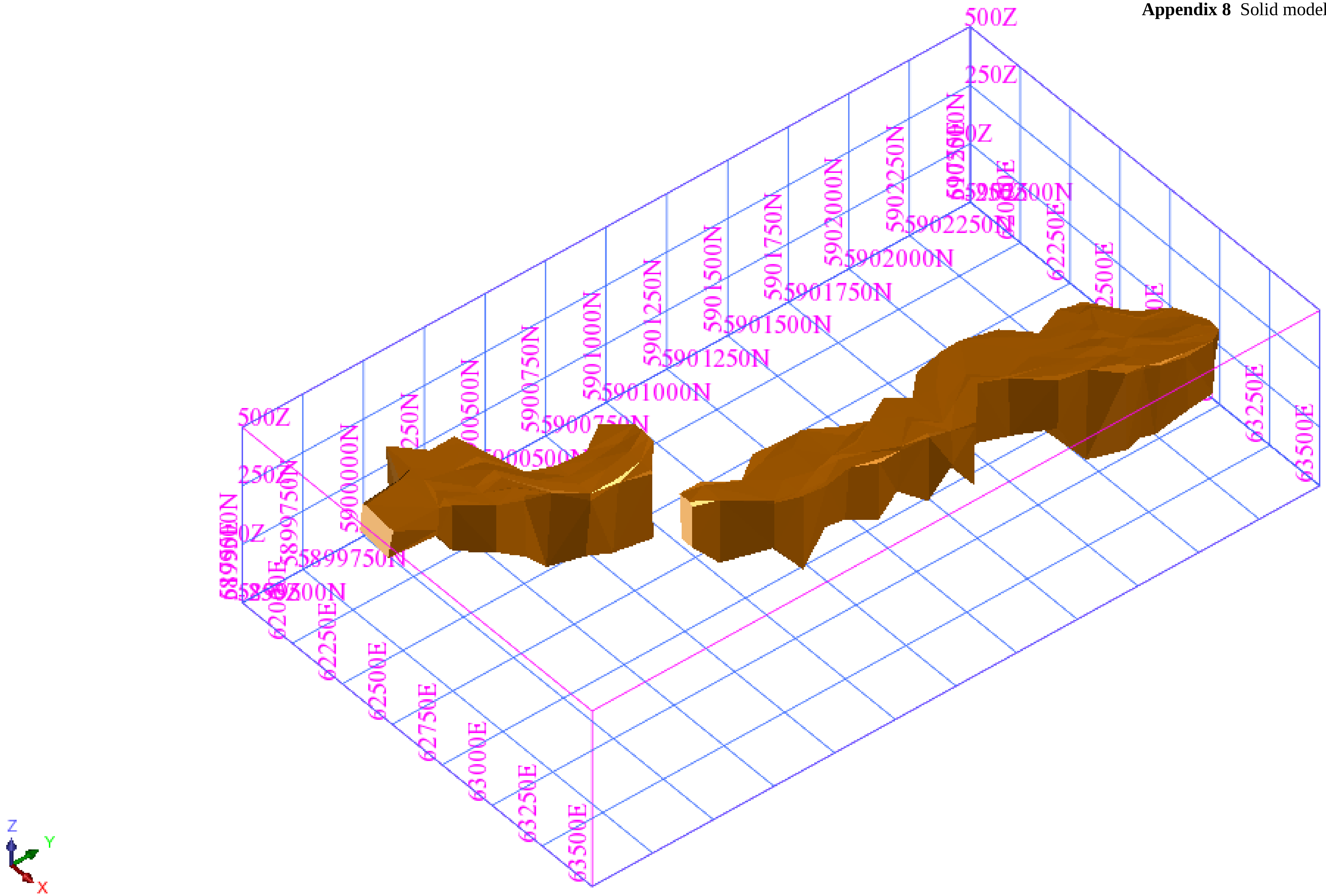
Appendix 6 Block model parameters.

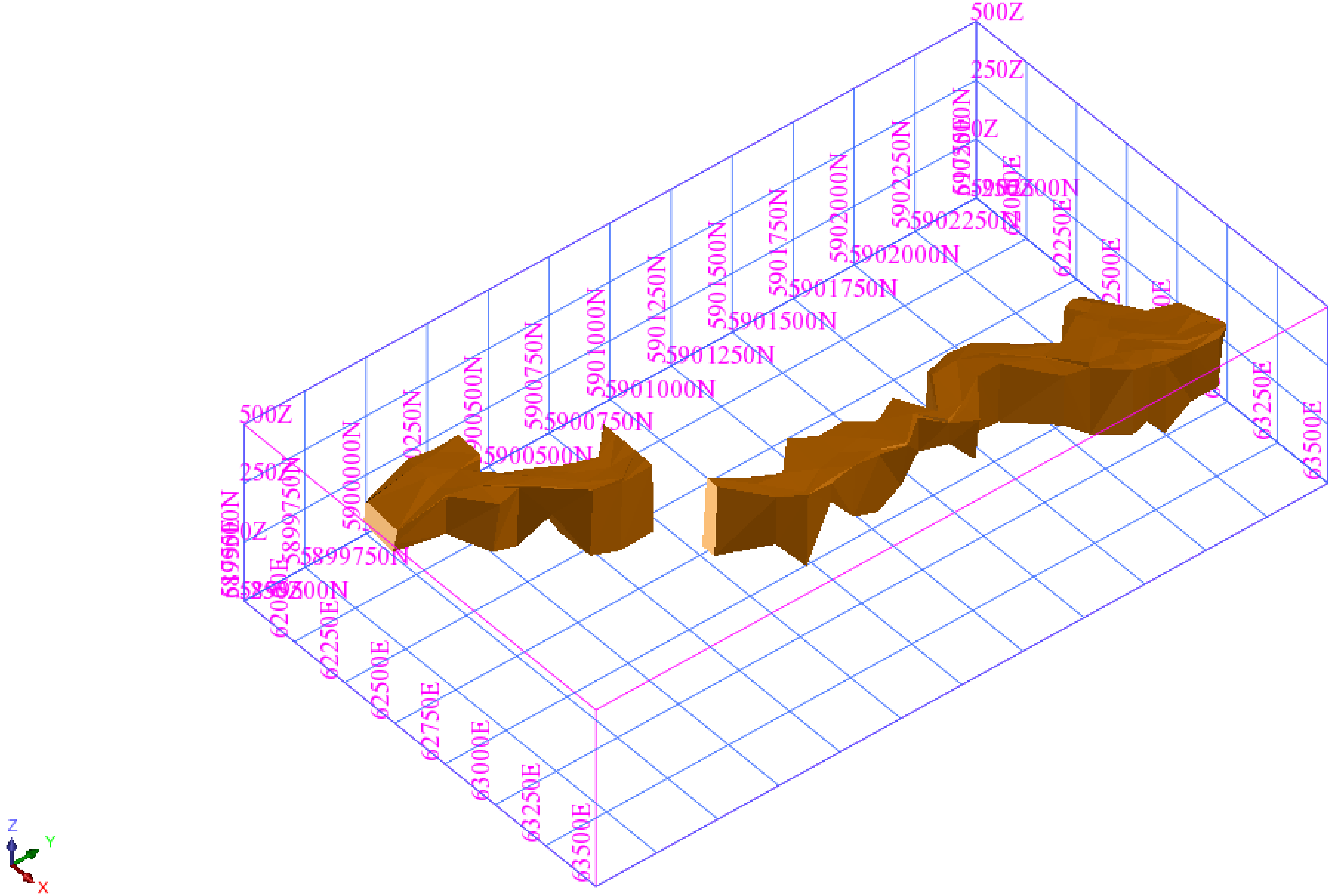
	Coordinates		
	Y	**X**	**Z**
Min coordinates	5 899 556	61 566	-198
Max coordinates	5 903 076	63 446	292
User block size	20	20	10
Min block size	20	20	10
	Rotation parameters		
	Bearing	**Dip**	**Plunge**
Rotation	0	0	0

Appendix 7 Solid models: 0,2% Cu grade

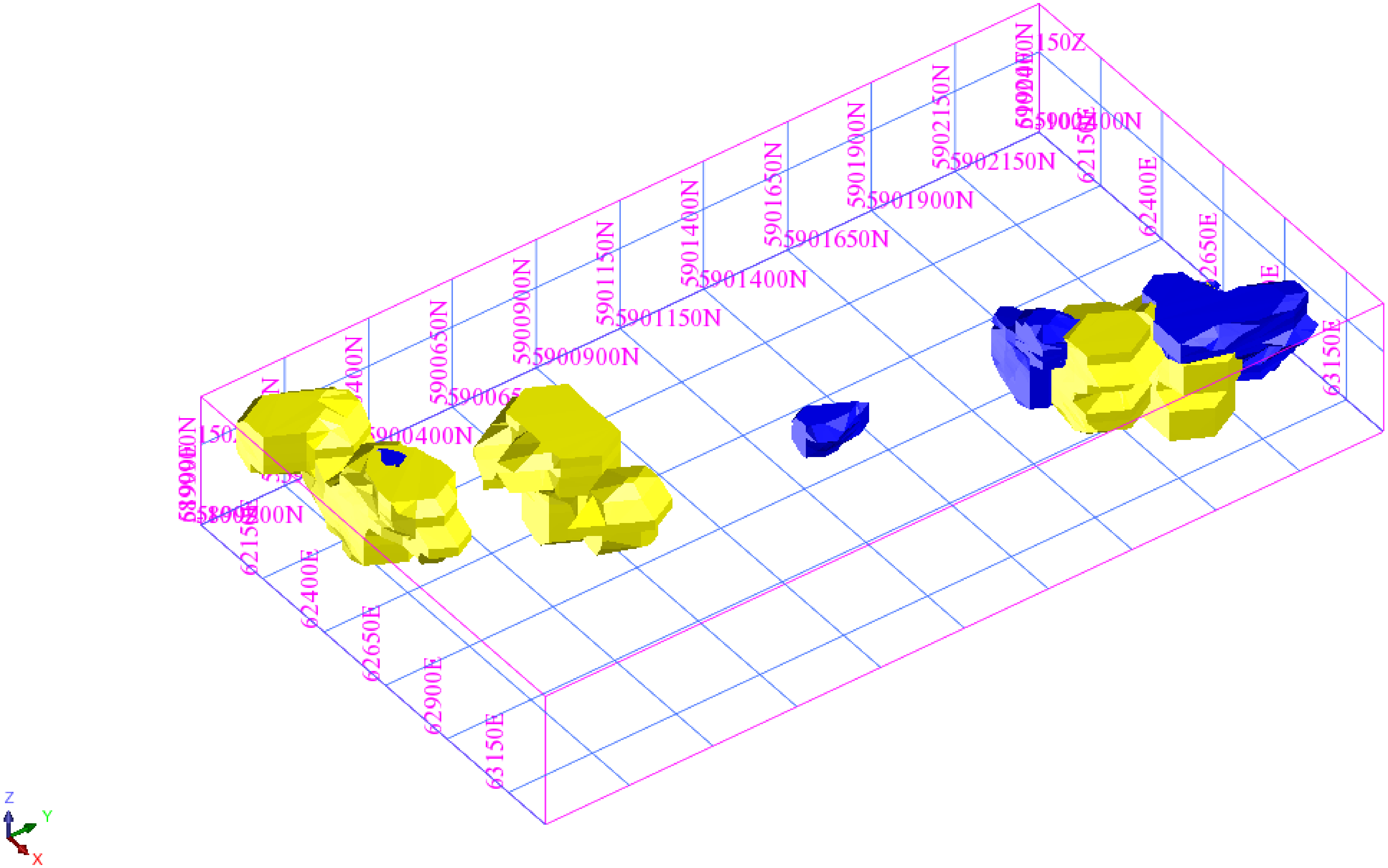


Appendix 8 Solid models: 0,3% Cu grade




500Z
250Z
5900000N
5900500N
5900750N
5901000N
5901250N
5901500N
5901750N
5902000N
5902250N
5899750N
62250E
62500E
62750E
63000E
63250E
63500E
Z
Y
X

Appendix 10 Solid models: magnetite (yellow) and bornite (blue) rich area



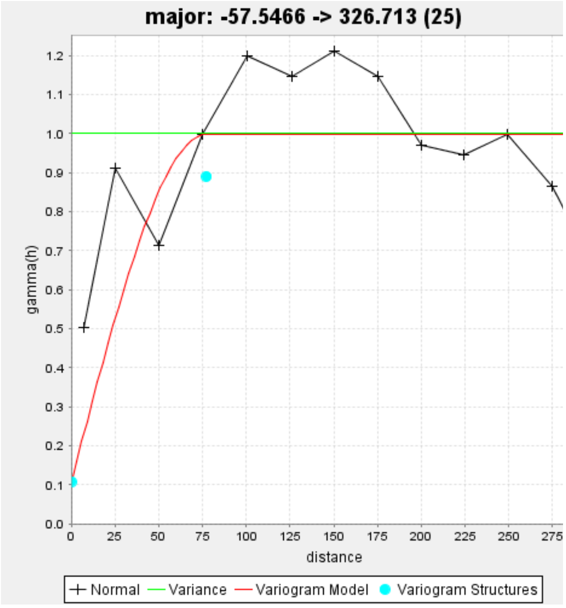


Figure A11-1 Major axis variogram (Cu)

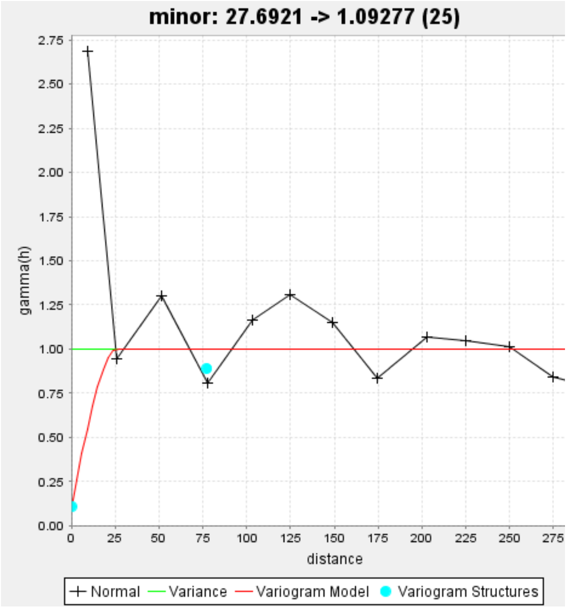


Figure A11-2 Minor axis variogram (Cu)

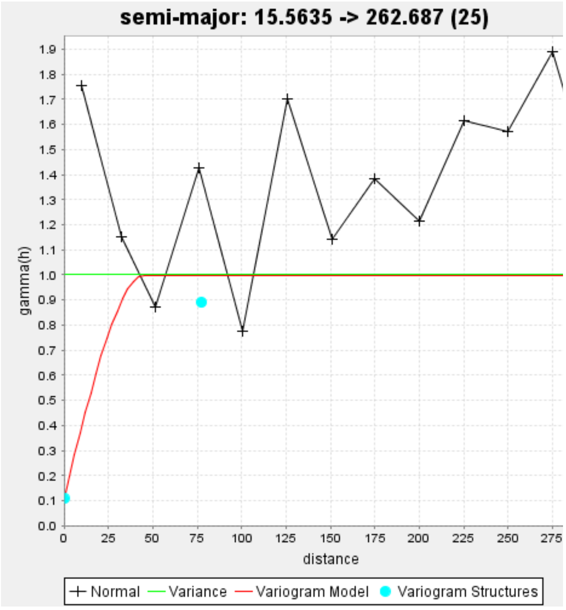


Figure A11-3 Semi-major axis variogram (Cu)

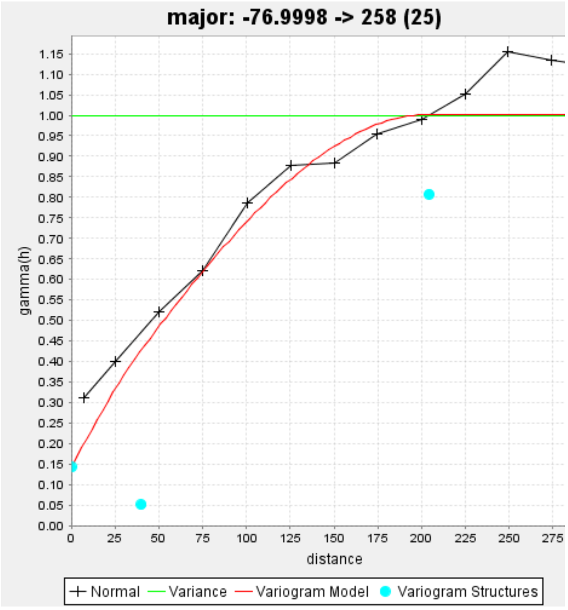


Figure A11-4 Major axis variogram (Au)

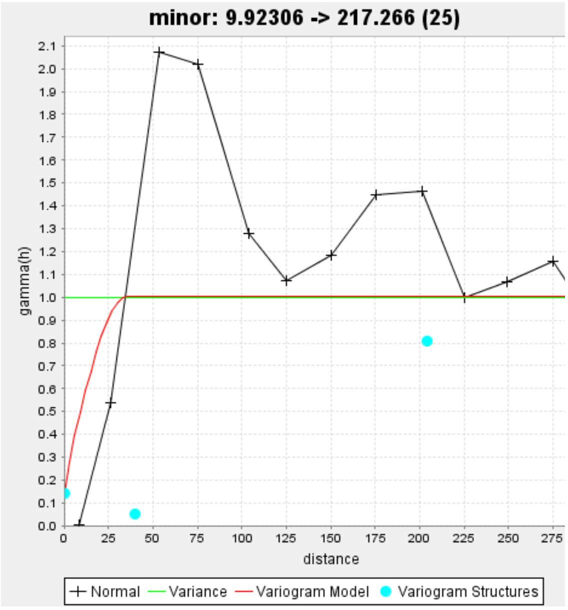


Figure A11-5 Minor axis variogram (Au)

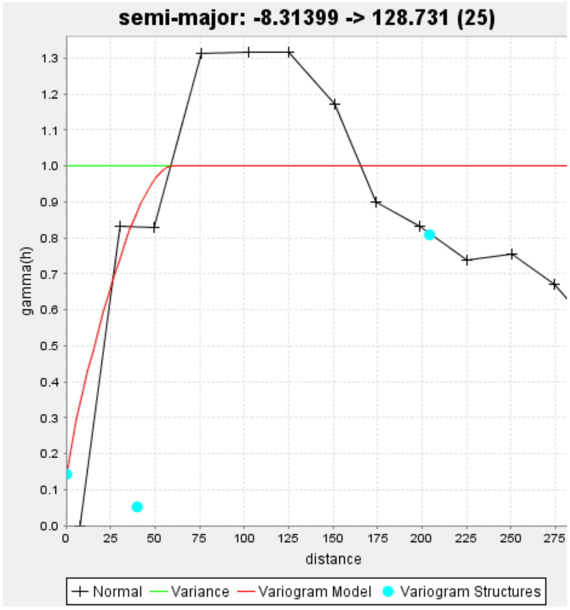


Figure A11-6 Semi-major axis variogram (Au)

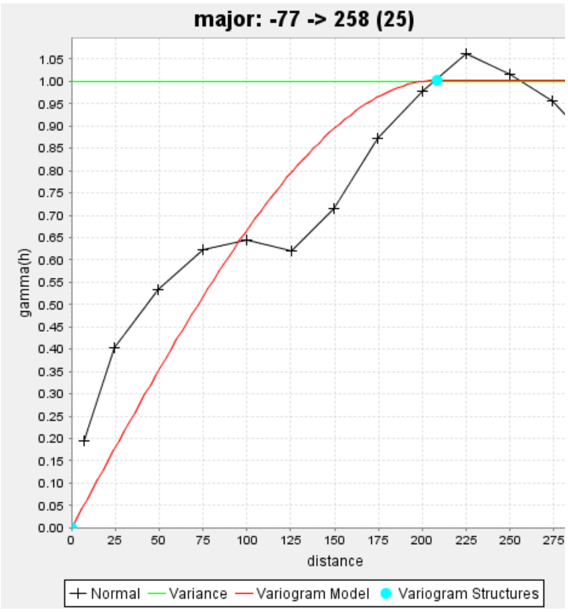


Figure A11-7 Major axis variogram (Hardness)

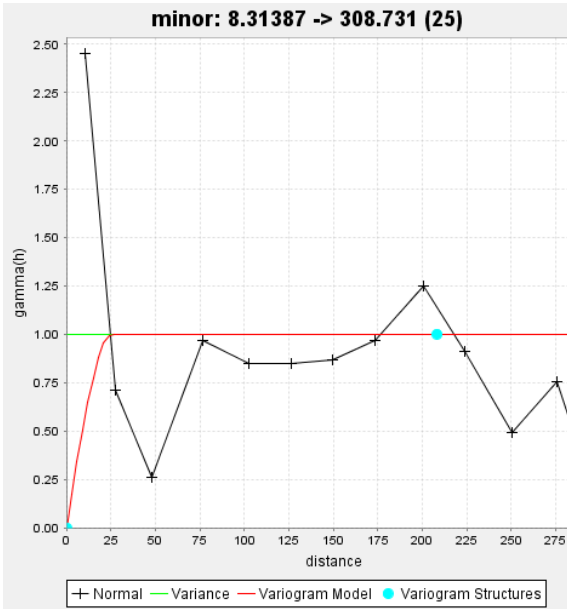


Figure A11-8 Minor axis variogram (Hardness)

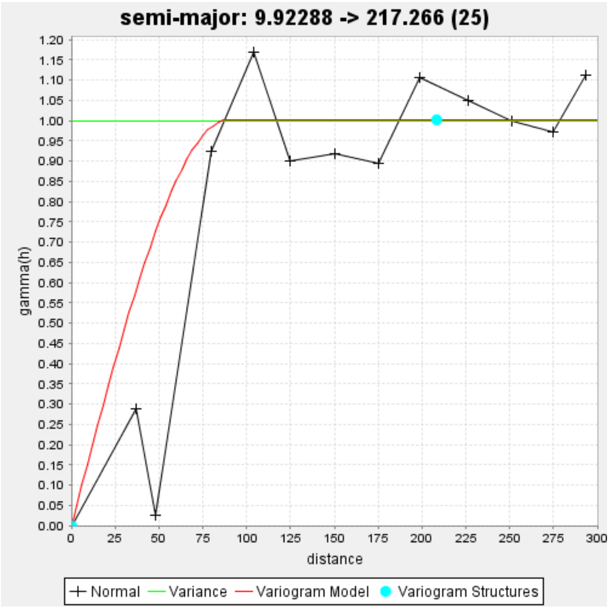
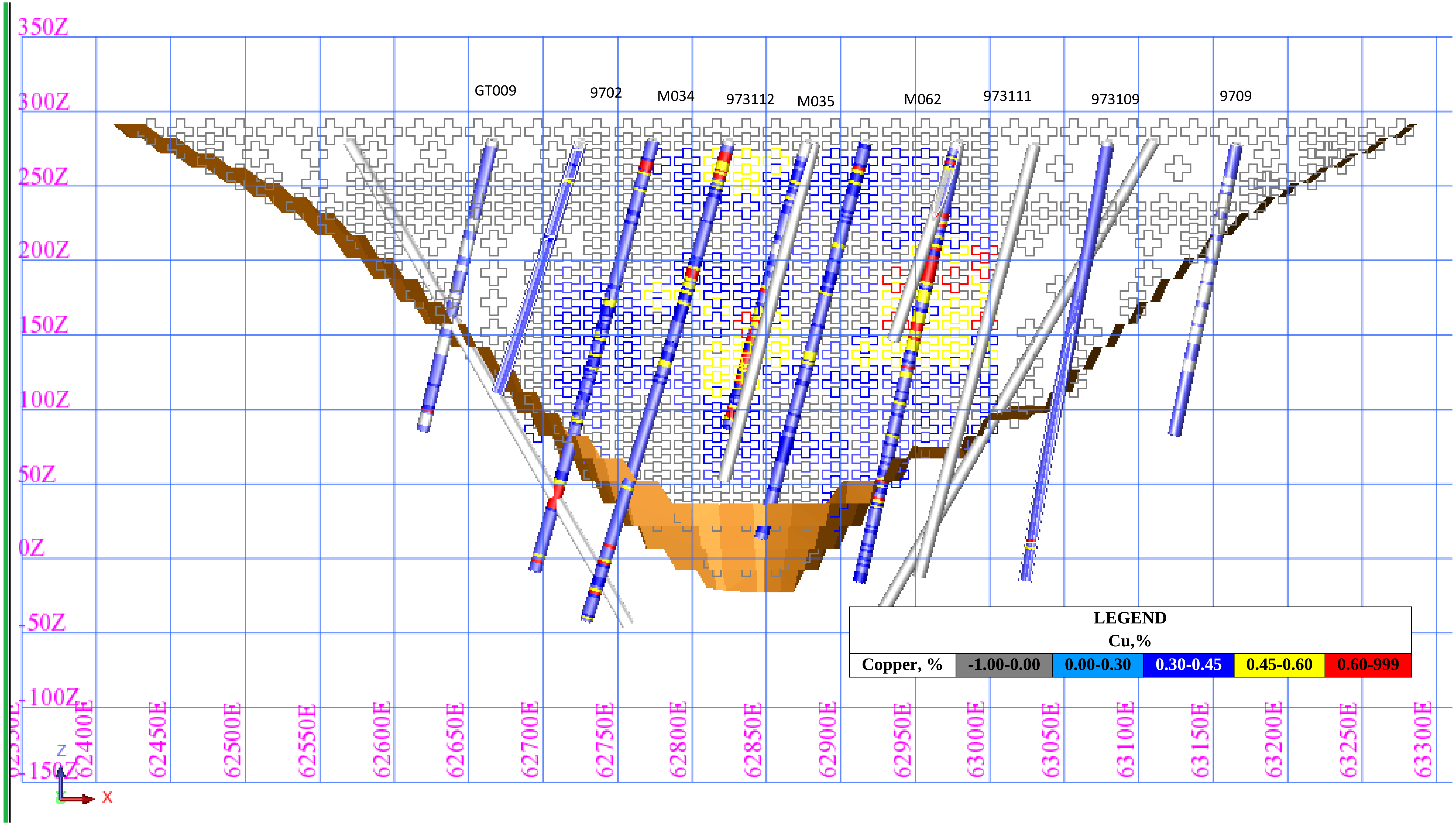
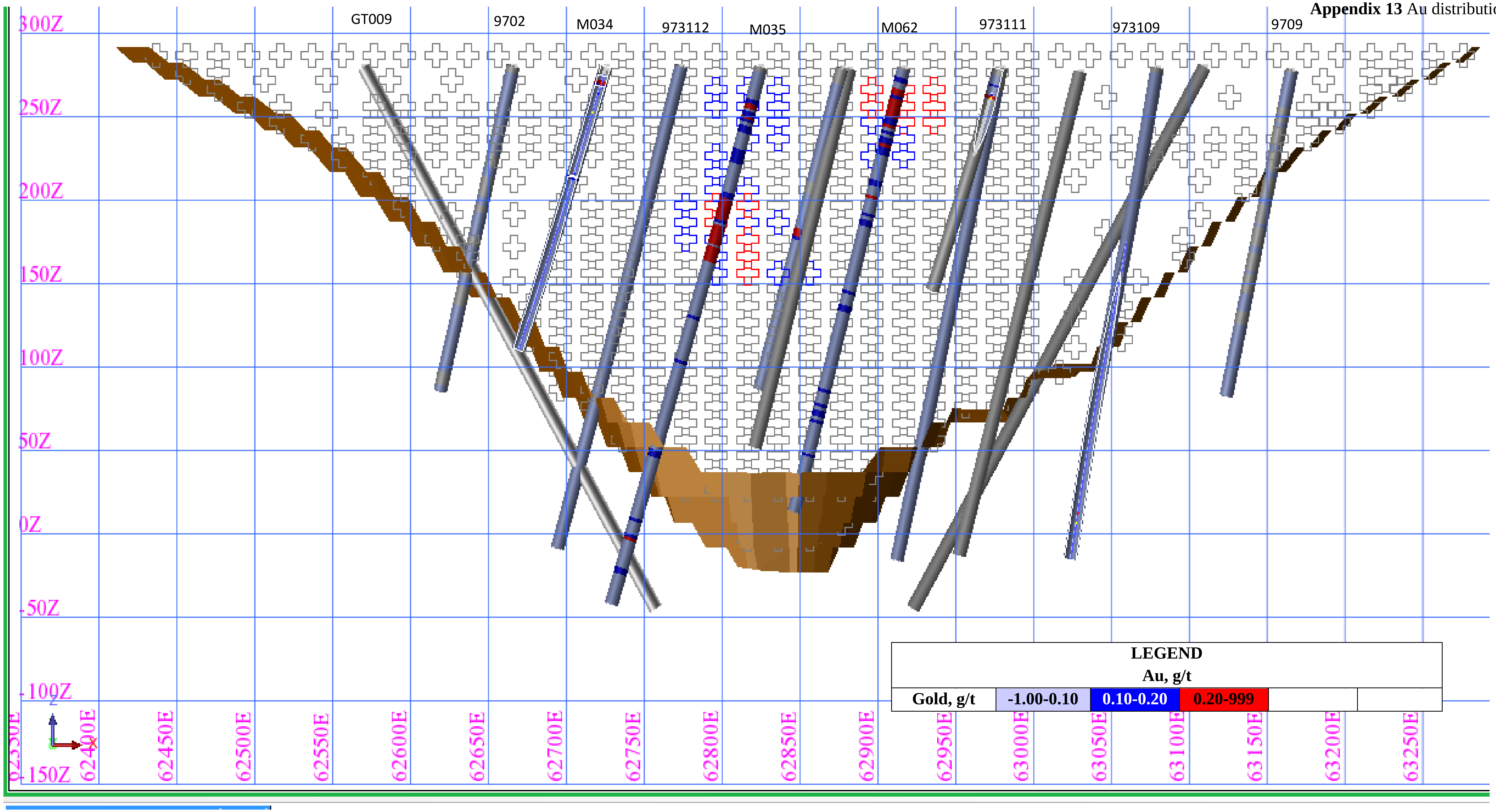


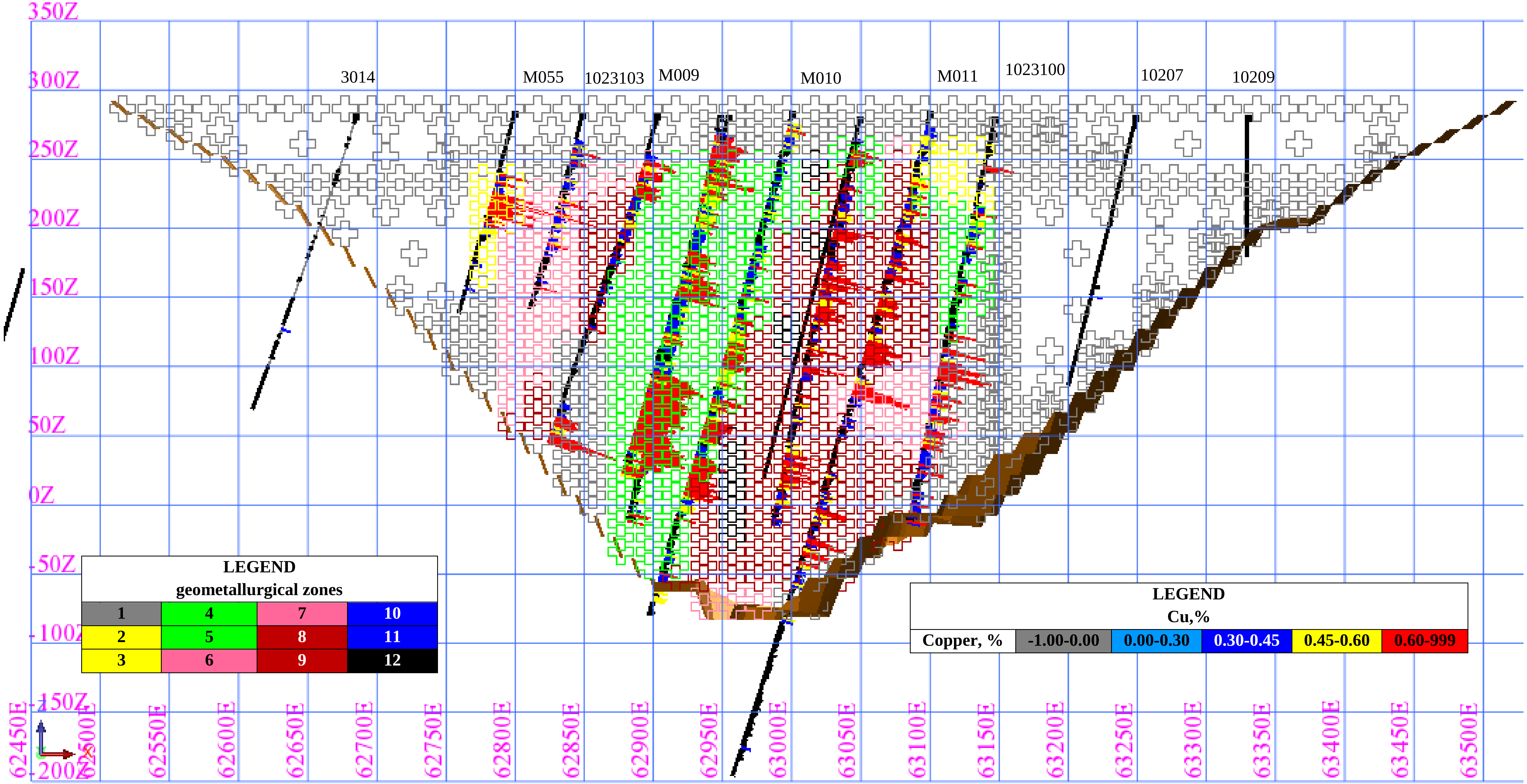
Figure A11-9 Semi-major axis variogram (Hardness)

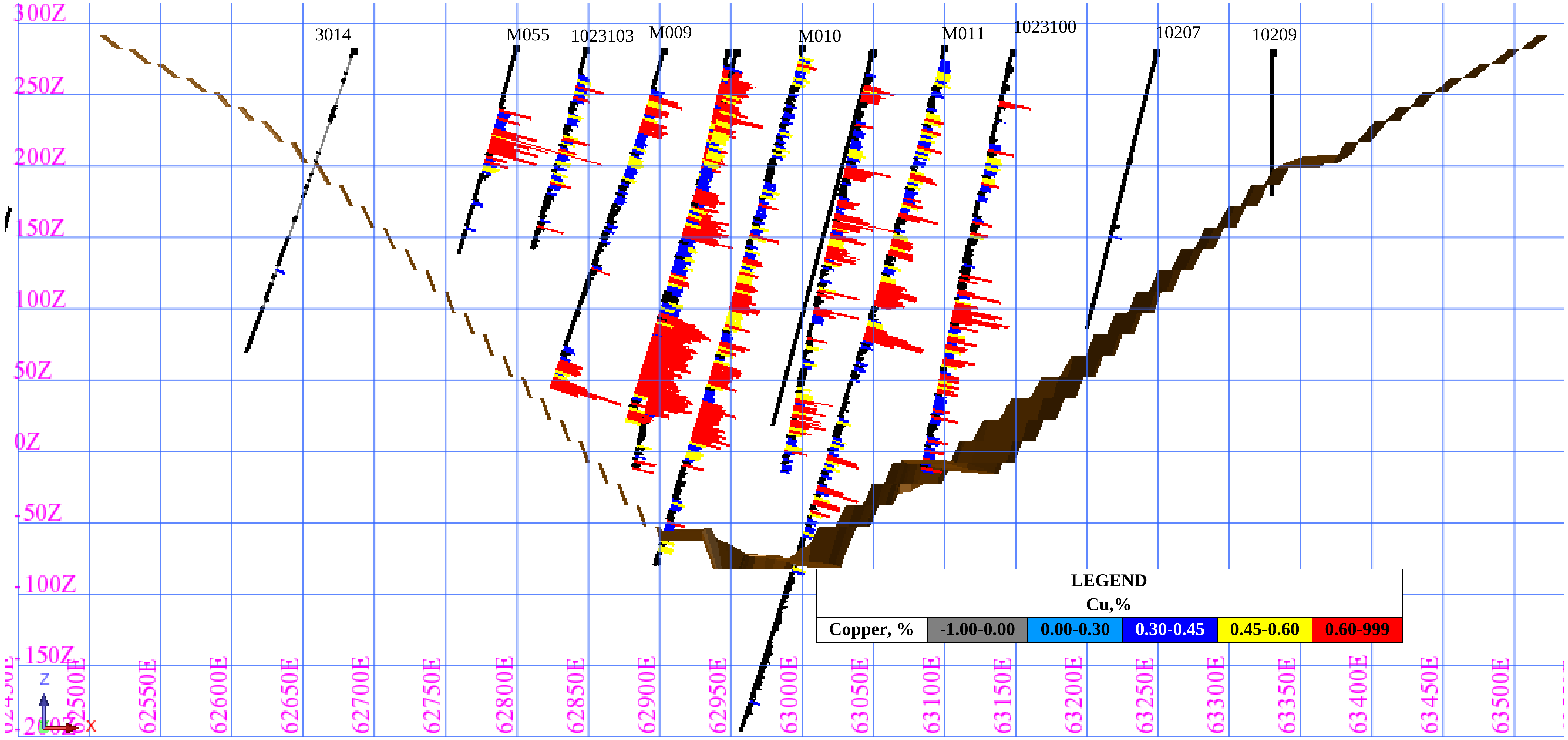

350Z
300Z
250Z
200Z
150Z
100Z
50Z
0Z
-50Z
-100Z
-150Z
GT009
9702
M034
973112
M035
M062
973111
973109
9709
62400E
62450E
62500E
62550E
62600E
62650E
62700E
62750E
62800E
62850E
62900E
62950E
63000E
63050E
63100E
63150E
63200E
63250E
63300E
Z
X
LEGEND
Cu,%
Copper, %
-1.00-0.00
0.00-0.30
0.30-0.45
0.45-0.60
0.60-999

Appendix 13 Au distribution



LEGEND					
Au, g/t					
Gold, g/t	-1.00-0.10	0.10-0.20	0.20-999		


3014
M055
1023103
M009
M010
M011
1023100
10207
10209
350Z
300Z
250Z
200Z
150Z
100Z
50Z
0Z
-50Z
-100Z
-150Z
-200Z
62450E
62500E
62550E
62600E
62650E
62700E
62750E
62800E
62850E
62900E
62950E
63000E
63050E
63100E
63150E
63200E
63250E
63300E
63350E
63400E
63450E
63500E
LEGEND
geometallurgical zones
1
2
3
4
5
6
7
8
9
10
11
12
LEGEND
Cu,%
Copper, %
-1.00-0.00
0.00-0.30
0.30-0.45
0.45-0.60
0.60-999


300Z
250Z
200Z
150Z
100Z
50Z
0Z
-50Z
-100Z
-150Z
3014
M055
1023103
M009
M010
M011
1023100
10207
10209
LEGEND
Cu,%
Copper, %
-1.00-0.00
0.00-0.30
0.30-0.45
0.45-0.60
0.60-999
62550E
62600E
62650E
62700E
62750E
62800E
62850E
62900E
62950E
63000E
63050E
63100E
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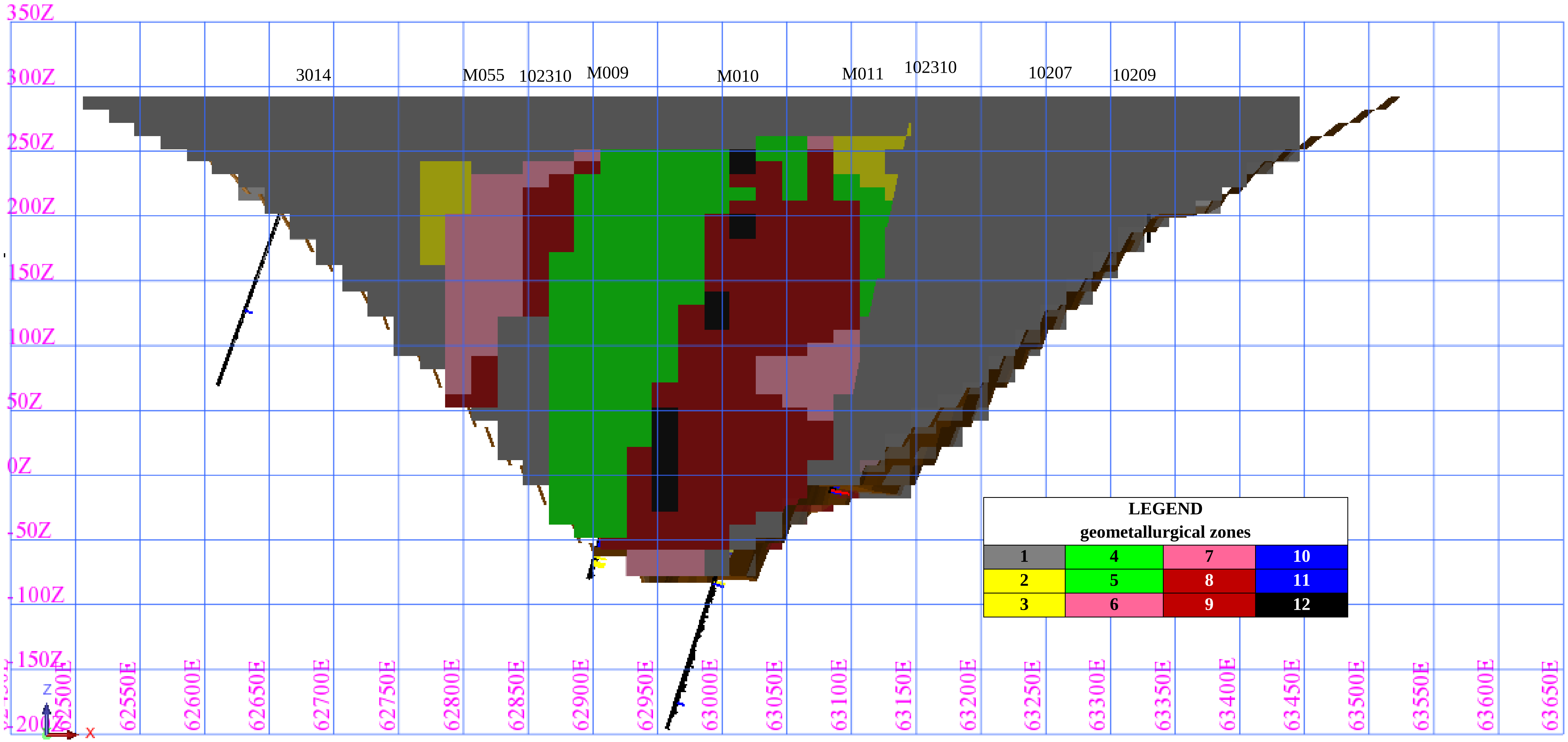
63300E
63350E
63400E
63450E
63500E
Z
X



LEGEND
geometallurgical zones

1	4	7	10
2	5	8	11
3	6	9	12

Table A17 - 1 Mine schedule, case scenario 1

Pit stage	*Material Name*	*Tonnes, kt*	*Cu, %*	*Au, g/t*	*I.1*	*II.1*	*III.1*	*IV.1*	*I.2*	*II.2*	*III.2*	*IV.2*	*I.3*	*II.3*	*III.3*	*IV.3*	*I.4*	*II.4*	*III.4*	*IV.4*	*I.5*	*II.5*	*III.5*	*IV.5*	*I.6*	*II.6*	*Control*
1	1	12,454	0.11	0.02	6,250	2,070	2,100	2,034																			**Waste**
1	3	72	0.53	0.12	72.0																						**12,454**
1	5	18	0.74	0.20		18																					**Oxides**
1	7	108	0.66	0.12		108																					**216**
1	9	9	0.67	-		9																					**Ore**
1	11	9	0.63	-		9																					**17,443**
1	2	3,694	0.56	0.07		3,694																					**Total**
1	4	572	0.59	0.10		572																					**30,112**
1	6	8,464	0.61	0.08			4,500	3,964																			
1	8	3,225	0.69	0.07				536	2,689																		
1	10	1,487	0.61	0.09					1,487																		
1	**Grand Total**	30,112	0.40	0.05	6,322	6,480	6,600	6,534	4,176																		
	Average, no waste		0.61	0.08																							
2	1	5,327	0.03	0.01				100	2,614	2,614																	**Waste**
2	3	9	0.75	0.09					9																		**5,327**
2	5	9	0.59	0.01					9																		**Oxides**
2	9	9	0.66	0.13					9																		**27**
2	2	2,219	0.41	0.04					297	1,922																	**Ore**
2	4	789	0.47	0.06						789																	**7,801**
2	6	2,505	0.54	0.07						1,788	716																**Total**
2	8	2,162	0.51	0.09							2,162																**13,155**
2	10	103	0.65	0.11							103																
2	12	23	0.49	0.04							23																
2	**Grand Total**	13,155	0.31	0.04				100	2,937	7,114	3,004																
	Average, no waste		0.49	0.06																							
3	1	13,843	0.08	0.01							2,750	2,750	2,750	2,750	2,750	93											**Waste**
3	3	9	0.27	0.05							9																**13,843**
3	5	36	0.30	0.02							36																**Oxides**
3	7	63	0.41	0.05							63																**171**
3	9	63	0.59	0.14							30	33															**Ore**
3	2	2,379	0.38	0.03									2,379														**29,189**
3	4	6,908	0.49	0.08							1,358	4,467	1,083														**Total**
3	6	6,085	0.36	0.05									1,038	4,500	547												**43,204**
3	8	11,907	0.41	0.08										-	3,953	4,500	3,454										
3	10	286	0.55	0.04													286										
3	12	1,624	0.45	0.04														1,624									
3	**Grand Total**	43,204	0.31	0.05							4,246	7,250	7,250	7,250	7,250	4,593	3,740	1,624									
	Average, no waste		0.42	0.07																							
4	1	13,883	0.03	0.00												2,657	2,800	2,800	2,800	2,800	26						**Waste**
4	7	45	0.27	0.01													45										**13,883**
4	2	2,253	0.31	0.03													715	1,538									**Oxides**
4	4	2,105	0.46	0.05														1,338	767								**45**
4	6	5,845	0.33	0.03														-	3,733	2,112							**Ore**
4	8	8,704	0.42	0.06																2,388	4,500	1,816					**19,833**
4	10	103	0.66	0.07																		103					**Total**
4	12	824	0.68	0.04																		824					**33,761**
4	**Grand Total**	33,761	0.25	0.03												2,657	3,560	5,676	7,300	7,300	4,527	2,742					
	Average, no waste		0.40	0.05																							
5	1	20,181	0.01	0.00																	3,174	3,400	3,600	3,800	4,000	2,207	**Waste**
5	3	9	0.16	-																		9					**20,181**
5	2	1,624	0.31	0.04																		1,624					**Oxides**
5	4	1,418	0.44	0.06																		124	1,294				**9**
5	6	7,469	0.38	0.03																			3,206	4,263			**Ore**
5	8	7,629	0.42	0.05																				237	4,500	2,892	**18,975**
5	10	320	0.52	0.07																						320	**Total**
5	12	515	0.59	0.04																						515	**39,165**
5	**Grand Total**	39,165	0.20	0.02																							
	Average, no waste		0.40	0.04																							
	Grand total	159,397	0.4533	0.06																							
waste, kt					6,250	2,070	2,100	2,134	2,614	2,614	2,750	2,750	2,750	2,750	2,750	2,750	2,800	2,800	2,800	2,800	3,200	3,400	3,600	3,800	4,000	2,207	**65,689**
ore, kt					-	4,266	4,500	4,500	4,473	4,500	4,362	4,467	4,500	4,500	4,500	4,500	4,455	4,500	4,500	4,500	4,500	4,491	4,500	4,500	4,500	3,727	**93,241**
transitional, kt					72	144	-	-	27	-	138	33	-	-	-	-	45	-	-	-	-	9	-	-	-	-	**468**
ore+oxides, kt					72	4,410	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	3,727	**93,709**
total, kt					6,322	6,480	6,600	6,634	7,114	7,114	7,250	7,250	7,250	7,250	7,250	7,250	7,300	7,300	7,300	7,300	7,700	7,900	8,100	8,300	8,500	5,934	**159,397**

Table A17 - 2 Cu grades, case scenario 1

Pit stage	*Material Name*	*Tonnes, kt*	*Cu, %*	*Au, g/t*	*I.1*	*II.1*	*III.1*	*IV.1*	*I.2*	*II.2*	*III.2*	*IV.2*	*I.3*	*II.3*	*III.3*	*IV.3*	*I.4*	*II.4*	*III.4*	*IV.4*	*I.5*	*II.5*	*III.5*	*IV.5*	*I.6*	*II.6*	*Control*
1	1	12,454	0.11	0.02																							
1	3	72	0.53	0.12	0.53	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	5	18	0.74	0.20	-	0.74	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	7	108	0.66	0.12	-	0.66	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	9	9	0.67	-	-	0.67	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	11	9	0.63	-	-	0.63	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	2	3,694	0.56	0.07	-	0.56	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	4	572	0.59	0.10	-	0.59	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	6	8,464	0.61	0.08	-	-	0.61	0.61	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	8	3,225	0.69	0.07	-	-	-	0.69	0.69	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	10	1,487	0.61	0.09	-	-	-	-	0.61	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	**Grand Total**	30,112	0.40	0.05																							
	Average, no waste		0.61	0.08																							
2	1	5,327	0.03	0.01																							
2	3	9	0.75	0.09	-	-	-	-	0.75	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	5	9	0.59	0.01	-	-	-	-	0.59	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	9	9	0.66	0.13	-	-	-	-	0.66	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	2	2,219	0.41	0.04	-	-	-	-	0.41	0.41	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	4	789	0.47	0.06	-	-	-	-	-	0.47	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	6	2,505	0.54	0.07	-	-	-	-	-	0.54	0.54	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	8	2,162	0.51	0.09	-	-	-	-	-	-	0.51	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	10	103	0.65	0.11	-			-	-	-	0.65																
2	12	23	0.49	0.04	-			-	-	-	0.49																
2	**Grand Total**	13,155	0.31	0.04																							
	Average, no waste		0.49	0.06																							
3	1	13,843	0.08	0.01																							
3	3	9	0.27	0.05	-	-	-	-	-	-	0.27	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
3	5	36	0.30	0.02	-	-	-	-	-	-	0.30	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
3	7	63	0.41	0.05	-	-	-	-	-	-	0.41	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
3	9	63	0.59	0.14	-	-	-	-	-	-	0.59	0.59	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
3	2	2,379	0.38	0.03	-	-	-	-	-	-	-	-	0.38	-	-	-	-	-	-	-	-	-	-	-	-	-	
3	4	6,908	0.49	0.08	-	-	-	-	-	-	0.49	0.49	0.49	-	-	-	-	-	-	-	-	-	-	-	-	-	
3	6	6,085	0.36	0.05	-	-	-	-	-	-	-	-	0.36	0.36	0.36	-	-	-	-	-	-	-	-	-	-	-	
3	8	11,907	0.41	0.08	-	-	-	-	-	-	-	-	-	-	0.41	0.41	0.41	-	-	-	-	-	-	-	-	-	
3	10	286	0.55	0.04	-	-	-	-	-	-	-	-	-	-	-	-	0.55	-	-	-	-	-	-	-	-	-	
3	12	1,624	0.45	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	0.45	-	-	-	-	-	-	-	-	
3	**Grand Total**	43,204	0.31	0.05																							
	Average, no waste		0.42	0.07																							
4	1	13,883	0.03	0.00																							
4	7	45	0.27	0.01	-	-	-	-	-	-	-	-	-	-	-	-	0.27	-	-	-	-	-	-	-	-	-	
4	2	2,253	0.31	0.03	-	-	-	-	-	-	-	-	-	-	-	-	0.31	0.31	-	-	-	-	-	-	-	-	
4	4	2,105	0.46	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	0.46	0.46	-	-	-	-	-	-	-	
4	6	5,845	0.33	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.33	0.33	-	-	-	-	-	-	
4	8	8,704	0.42	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.42	0.42	0.42	-	-	-	-	
4	10	103	0.66	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.66	-	-	-	-	
4	12	824	0.68	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.68	-	-	-	-	
4	**Grand Total**	33,761	0.25	0.03																							
	Average, no waste		0.40	0.05																							
5	1	20,181	0.01	0.00																							
5	3	9	0.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.16	-	-	-	-	
5	2	1,624	0.31	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.31	-	-	-	-	
5	4	1,418	0.44	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.44	0.44	-	-	-	
5	6	7,469	0.38	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.38	0.38	-	-	
5	8	7,629	0.42	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.42	0.42	0.42	
5	10	320	0.52	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.52	
5	12	515	0.59	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.59	
5	**Grand Total**	39,165	0.20	0.02																							
	Average, no waste		0.40	0.04																							
	GrandGrand total		-	0.06																							
grand total		159,397	0.4533	0.06																							
				average, %	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	
				max, %	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	**0.60**	
				real, %	**0.53**	**0.57**	**0.61**	**0.62**	**0.65**	**0.47**	**0.51**	**0.49**	**0.40**	**0.36**	**0.40**	**0.41**	**0.40**	**0.41**	**0.35**	**0.38**	**0.42**	**0.43**	**0.39**	**0.38**	**0.42**	**0.45**	**0.4533**
				min, %	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**	

Table A17 - 3 Cu tonnage, case scenario 1

Pit stage	*Material Name*	*Tonnes, kt*	*Cu, %*	*Au, g/t*	*I.1*	*II.1*	*III.1*	*IV.1*	*I.2*	*II.2*	*III.2*	*IV.2*	*I.3*	*II.3*	*III.3*	*IV.3*	*I.4*	*II.4*	*III.4*	*IV.4*	*I.5*	*II.5*	*III.5*	*IV.5*	*I.6*	*II.6*	*Control*
1	1	12,454	0.11	0.02																							**Waste**
1	3	72	0.53	0.12	0.32	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1	5	18	0.74	0.20	-	0.11	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Oxides**
1	7	108	0.66	0.12	-	0.60	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**1.14**
1	9	9	0.67	-	-	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Ore**
1	11	9	0.63	-	-	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**90.86**
1	2	3,694	0.56	0.07	-	17.48	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Total**
1	4	572	0.59	0.10	-	2.89	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**92.00**
1	6	8,464	0.61	0.08	-	-	23.24	20.48	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	8	3,225	0.69	0.07	-	-	-	3.16	15.87	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	10	1,487	0.61	0.09	-	-	-	-	7.74	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	**Grand Total**	30,112	0.40	0.05																							
	Average, no waste		0.61	0.08																							
2	1	5,327	0.03	0.01																							**Waste**
2	3	9	0.75	0.09	-	-	-	-	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
2	5	9	0.59	0.01	-	-	-	-	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Oxides**
2	9	9	0.66	0.13	-	-	-	-	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.15**
2	2	2,219	0.41	0.04	-	-	-	-	1.05	6.77	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Ore**
2	4	789	0.47	0.06	-	-	-	-	-	3.17	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**32.61**
2	6	2,505	0.54	0.07	-	-	-	-	-	8.22	3.29	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Total**
2	8	2,162	0.51	0.09	-	-	-	-	-	-	9.44	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**32.76**
2	10	103	0.65	0.11	-	-	-	-	-	-	0.57	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	12	23	0.49	0.04	-	-	-	-	-	-	0.09	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	**Grand Total**	13,155	0.31	0.04																							
	Average, no waste		0.49	0.06																							
3	1	13,843	0.08	0.01																							**Waste**
3	3	9	0.27	0.05	-	-	-	-	-	-	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
3	5	36	0.30	0.02	-	-	-	-	-	-	0.09	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Oxides**
3	7	63	0.41	0.05	-	-	-	-	-	-	0.22	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.65**
3	9	63	0.59	0.14	-	-	-	-	-	-	0.15	0.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Ore**
3	2	2,379	0.38	0.03	-	-	-	-	-	-	-	-	7.64	-	-	-	-	-	-	-	-	-	-	-	-	-	**97.60**
3	4	6,908	0.49	0.08	-	-	-	-	-	-	5.68	18.69	4.53	-	-	-	-	-	-	-	-	-	-	-	-	-	**Total**
3	6	6,085	0.36	0.05	-	-	-	-	-	-	-	-	3.18	13.79	1.68	-	-	-	-	-	-	-	-	-	-	-	**98.25**
3	8	11,907	0.41	0.08	-	-	-	-	-	-	-	-	-	-	13.64	15.53	11.92	-	-	-	-	-	-	-	-	-	
3	10	286	0.55	0.04	-	-	-	-	-	-	-	-	-	-	-	-	1.33	-	-	-	-	-	-	-	-	-	
3	12	1,624	0.45	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	6.23	-	-	-	-	-	-	-	-	
3	**Grand Total**	43,204	0.31	0.05																							
	Average, no waste		0.42	0.07																							
4	1	13,883	0.03	0.00																							**Waste**
4	7	45	0.27	0.01	-	-	-	-	-	-	-	-	-	-	-	-	0.10	-	-	-	-	-	-	-	-	-	-
4	2	2,253	0.31	0.03	-	-	-	-	-	-	-	-	-	-	-	-	1.91	4.11	-	-	-	-	-	-	-	-	**Oxides**
4	4	2,105	0.46	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	5.24	3.00	-	-	-	-	-	-	-	**0.10**
4	6	5,845	0.33	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	10.38	5.87	-	-	-	-	-	-	**Ore**
4	8	8,704	0.42	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	8.51	16.04	6.47	-	-	-	-	**66.90**
4	10	103	0.66	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.57	-	-	-	-	**Total**
4	12	824	0.68	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4.78	-	-	-	-	**67.00**
4	**Grand Total**	33,761	0.25	0.03																							
	Average, no waste		0.40	0.05																							
5	1	20,181	0.01	0.00																							**Waste**
5	3	9	0.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.01	-	-	-	-	-
5	2	1,624	0.31	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4.30	-	-	-	-	**Oxides**
5	4	1,418	0.44	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.46	4.80	-	-	-	**0.01**
5	6	7,469	0.38	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	10.30	13.70	-	-	**Ore**
5	8	7,629	0.42	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.85	16.06	10.32	**64.79**
5	10	320	0.52	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.41	**Total**
5	12	515	0.59	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2.60	**64.80**
5	**Grand Total**	39,165	0.20	0.02																							
	Average, no waste		0.40	0.04																							
	GrandGrand total		-	0.06																							
grand total		159,397	0.4533	0.06																							
			Metal (Cu),	**kt**	**0.32**	**21.19**	**23.24**	**23.64**	**24.80**	**18.16**	**19.57**	**18.85**	**15.35**	**13.79**	**15.32**	**15.53**	**15.26**	**15.59**	**13.39**	**14.38**	**16.04**	**16.61**	**15.10**	**14.55**	**16.06**	**14.32**	**361.05**
			Concentrate,	**kt**	**1.48**	**97.21**	**106.63**	**108.43**	**113.77**	**83.32**	**89.75**	**86.47**	**70.43**	**63.26**	**70.25**	**71.22**	**70.01**	**71.50**	**61.41**	**65.98**	**73.56**	**76.18**	**69.27**	**66.72**	**73.66**	**65.70**	**1,656.20**

Table A17 - 4 Au grades, case scenario 1

Pit stage	*Material Name*	*Tonnes, kt*	*Cu, %*	*Au, g/t*	*I.1*	*II.1*	*III.1*	*IV.1*	*I.2*	*II.2*	*III.2*	*IV.2*	*I.3*	*II.3*	*III.3*	*IV.3*	*I.4*	*II.4*	*III.4*	*IV.4*	*I.5*	*II.5*	*III.5*	*IV.5*	*I.6*	*II.6*	*Control*
1	1	12,454	0.11	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Waste**
1	3	72	0.53	0.12	0.12	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
1	5	18	0.74	0.20	-	0.20	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Oxides**
1	7	108	0.66	0.12	-	0.12	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.11**
1	9	9	0.67	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Ore**
1	11	9	0.63	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.08**
1	2	3,694	0.56	0.07	-	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Total**
1	4	572	0.59	0.10	-	0.10	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.08**
1	6	8,464	0.61	0.08	-	-	0.08	0.08	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	8	3,225	0.69	0.07	-	-	-	0.07	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	10	1,487	0.61	0.09	-	-	-	-	0.09	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	**Grand Total**	30,112	0.40	0.05																							
	Average, no waste		0.61	0.08																							
2	1	5,327	0.03	0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Waste**
2	3	9	0.75	0.09	-	-	-	-	0.09	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
2	5	9	0.59	0.01	-	-	-	-	0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Oxides**
2	9	9	0.66	0.13	-	-	-	-	0.13	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.08**
2	2	2,219	0.41	0.04	-	-	-	-	0.04	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Ore**
2	4	789	0.47	0.06	-	-	-	-	-	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.06**
2	6	2,505	0.54	0.07	-	-	-	-	-	0.07	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Total**
2	8	2,162	0.51	0.09	-	-	-	-	-	-	0.09	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.06**
2	10	103	0.65	0.11	-	-	-	-	-	-	0.11	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	12	23	0.49	0.04	-	-	-	-	-	-	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	**Grand Total**	13,155	0.31	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	Average, no waste		0.49	0.06																							
3	1	13,843	0.08	0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Waste**
3	3	9	0.27	0.05	-	-	-	-	-	-	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
3	5	36	0.30	0.02	-	-	-	-	-	-	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Oxides**
3	7	63	0.41	0.05	-	-	-	-	-	-	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.08**
3	9	63	0.59	0.14	-	-	-	-	-	-	0.14	0.14	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Ore**
3	2	2,379	0.38	0.03	-	-	-	-	-	-	-	-	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.06**
3	4	6,908	0.49	0.08	-	-	-	-	-	-	0.08	0.08	0.08	-	-	-	-	-	-	-	-	-	-	-	-	-	**Total**
3	6	6,085	0.36	0.05	-	-	-	-	-	-	-	-	0.05	0.05	0.05	-	-	-	-	-	-	-	-	-	-	-	**0.06**
3	8	11,907	0.41	0.08	-	-	-	-	-	-	-	-	-	-	0.08	0.08	0.08	-	-	-	-	-	-	-	-	-	
3	10	286	0.55	0.04	-	-	-	-	-	-	-	-	-	-	-	-	0.04	-	-	-	-	-	-	-	-	-	
3	12	1,624	0.45	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	0.04	-	-	-	-	-	-	-	-	
3	**Grand Total**	43,204	0.31	0.05																							
	Average, no waste		0.42	0.07																							
4	1	13,883	0.03	0.00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Waste**
4	7	45	0.27	0.01	-	-	-	-	-	-	-	-	-	-	-	-	0.01	-	-	-	-	-	-	-	-	-	**-**
4	2	2,253	0.31	0.03	-	-	-	-	-	-	-	-	-	-	-	-	0.03	0.03	-	-	-	-	-	-	-	-	**Oxides**
4	4	2,105	0.46	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	0.05	0.05	-	-	-	-	-	-	-	**0.01**
4	6	5,845	0.33	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.03	0.03	-	-	-	-	-	-	**Ore**
4	8	8,704	0.42	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.06	0.06	0.06	-	-	-	-	**0.04**
4	10	103	0.66	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.07	-	-	-	-	**Total**
4	12	824	0.68	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.04	-	-	-	-	**0.04**
4	**Grand Total**	33,761	0.25	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	Average, no waste		0.40	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
5	1	20,181	0.01	0.00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**Waste**
5	3	9	0.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**-**
5	2	1,624	0.31	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.04	-	-	-	-	**Oxides**
5	4	1,418	0.44	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.06	0.06	-	-	-	**-**
5	6	7,469	0.38	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.03	0.03	-	-	**Ore**
5	8	7,629	0.42	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.05	0.05	0.05	**0.04**
5	10	320	0.52	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.07	**Total**
5	12	515	0.59	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.04	**0.04**
5	**Grand Total**	39,165	0.20	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	Average, no waste		0.40	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	GrandGrand total		-	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
grand total		159,397	0.4533	0.06																							
			Average	**Au, g/t**	**0.12**	**0.08**	**0.08**	**0.08**	**0.07**	**0.05**	**0.08**	**0.08**	**0.05**	**0.05**	**0.08**	**0.08**	**0.07**	**0.04**	**0.03**	**0.05**	**0.06**	**0.05**	**0.04**	**0.03**	**0.05**	**0.05**	**0.06**

Table A17 - 5 Au masses, case scenario 1

Pit stage	*Material Name*	*Tonnes, kt*	*Cu, %*	*Au, g/t*	*I.1*	*II.1*	*III.1*	*IV.1*	*I.2*	*II.2*	*III.2*	*IV.2*	*I.3*	*II.3*	*III.3*	*IV.3*	*I.4*	*II.4*	*III.4*	*IV.4*	*I.5*	*II.5*	*III.5*	*IV.5*	*I.6*	*II.6*	*Control*
1	1	12,454	0.11	0.02																							Waste
1	3	72	0.53	0.12	5.56	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1	5	18	0.74	0.20	-	2.33	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Oxides
1	7	108	0.66	0.12	-	8.23	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	16.12
1	9	9	0.67	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Ore
1	11	9	0.63	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	896.46
1	2	3,694	0.56	0.07	-	170.81	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Total
1	4	572	0.59	0.10	-	35.91	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	912.58
1	6	8,464	0.61	0.08	-	-	243.11	214.15	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	8	3,225	0.69	0.07	-	-	-	23.89	119.86	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	10	1,487	0.61	0.09	-	-	-	-	88.73	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	**Grand Total**	30,112	0.40	0.05																							
	Average, no waste		0.61	0.08																							
2	1	5,327	0.03	0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
2	3	9	0.75	0.09	-	-	-	-	0.51	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
2	5	9	0.59	0.01	-	-	-	-	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Oxides
2	9	9	0.66	0.13	-	-	-	-	0.78	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.35
2	2	2,219	0.41	0.04	-	-	-	-	7.38	47.82	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Ore
2	4	789	0.47	0.06	-	-	-	-	-	30.36	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	329.16
2	6	2,505	0.54	0.07	-	-	-	-	-	81.29	32.56	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Total
2	8	2,162	0.51	0.09	-	-	-	-	-	-	122.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	330.51
2	10	103	0.65	0.11	-	-	-	-	-	-	7.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	12	23	0.49	0.04	-	-	-	-	-	-	0.59	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	**Grand Total**	13,155	0.31	0.04																							
	Average, no waste		0.49	0.06																							
3	1	13,843	0.08	0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
3	3	9	0.27	0.05	-	-	-	-	-	-	0.29	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
3	5	36	0.30	0.02	-	-	-	-	-	-	0.45	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Oxides
3	7	63	0.41	0.05	-	-	-	-	-	-	2.25	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	8.63
3	9	63	0.59	0.14	-	-	-	-	-	-	2.69	2.95	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Ore
3	2	2,379	0.38	0.03	-	-	-	-	-	-	-	-	49.95	-	-	-	-	-	-	-	-	-	-	-	-	-	1,227.82
3	4	6,908	0.49	0.08	-	-	-	-	-	-	72.71	239.16	58.00	-	-	-	-	-	-	-	-	-	-	-	-	-	Total
3	6	6,085	0.36	0.05	-	-	-	-	-	-	-	-	32.32	140.18	17.05	-	-	-	-	-	-	-	-	-	-	-	1,236.45
3	8	11,907	0.41	0.08	-	-	-	-	-	-	-	-	-	-	202.90	230.99	177.30	-	-	-	-	-	-	-	-	-	
3	10	286	0.55	0.04	-	-	-	-	-	-	-	-	-	-	-	-	7.25	-	-	-	-	-	-	-	-	-	
3	12	1,624	0.45	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	37.90	-	-	-	-	-	-	-	-	
3	**Grand Total**	43,204	0.31	0.05																							
	Average, no waste		0.42	0.07																							
4	1	13,883	0.03	0.00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
4	7	45	0.27	0.01	-	-	-	-	-	-	-	-	-	-	-	-	0.15	-	-	-	-	-	-	-	-	-	-
4	2	2,253	0.31	0.03	-	-	-	-	-	-	-	-	-	-	-	-	14.99	32.26	-	-	-	-	-	-	-	-	Oxides
4	4	2,105	0.46	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	44.32	25.42	-	-	-	-	-	-	-	0.15
4	6	5,845	0.33	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	73.57	41.62	-	-	-	-	-	-	Ore
4	8	8,704	0.42	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	90.87	171.22	69.09	-	-	-	-	589.60
4	10	103	0.66	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4.77	-	-	-	-	Total
4	12	824	0.68	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	21.45	-	-	-	-	589.74
4	**Grand Total**	33,761	0.25	0.03																							
	Average, no waste		0.40	0.05																							
5	1	20,181	0.01	0.00																							Waste
5	3	9	0.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
5	2	1,624	0.31	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	41.84	-	-	-	-	Oxides
5	4	1,418	0.44	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.04	52.37	-	-	-	-
5	6	7,469	0.38	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	69.40	92.27	-	-	Ore
5	8	7,629	0.42	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	7.44	141.15	90.70	529.24
5	10	320	0.52	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	15.24	Total
5	12	515	0.59	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	13.79	529.24
5	**Grand Total**	39,165	0.20	0.02																							
	Average, no waste		0.40	0.04																							
	GrandGrand total		-	0.06																							
grand total		159,397	0.4533	0.06																							
			Total	**Au, kg**	**5.56**	**217.29**	**243.11**	**238.04**	**217.32**	**159.47**	**240.70**	**242.11**	**140.27**	**140.18**	**219.95**	**230.99**	**199.70**	**114.48**	**98.99**	**132.49**	**171.22**	**142.19**	**121.77**	**99.71**	**141.15**	**119.73**	**3,636.42**

Table A17 - 6 Revenue, M$, case scenario 1

Pit stage	*Material Name*	*Tonnes, kt*	*Cu, %*	*Au, g/t*	*I.1*	*II.1*	*III.1*	*IV.1*	*I.2*	*II.2*	*III.2*	*IV.2*	*I.3*	*II.3*	*III.3*	*IV.3*	*I.4*	*II.4*	*III.4*	*IV.4*	*I.5*	*II.5*	*III.5*	*IV.5*	*I.6*	*II.6*	*Control*
1	1	12,454	0.11	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
1	3	72	0.53	0.12	1.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1	5	18	0.74	0.20	-	0.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Oxides
1	7	108	0.66	0.12	-	3.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	7
1	9	9	0.67	-	-	0.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Ore
1	11	9	0.63	-	-	0.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	522
1	2	3,694	0.56	0.07	-	100.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Total
1	4	572	0.59	0.10	-	16.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	528
1	6	8,464	0.61	0.08	-	-	133.5	117.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	8	3,225	0.69	0.07	-	-	-	18.2	91.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	10	1,487	0.61	0.09	-	-	-	-	44.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	**GrandTotal**	30,112	0.40	0.05	1.8	121.7	133.5	135.7	135.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	Average,nowaste		0.61	0.08																							
2	1	5,327	0.03	0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
2	3	9	0.75	0.09	-	-	-	-	0.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
2	5	9	0.59	0.01	-	-	-	-	0.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Oxides
2	9	9	0.66	0.13	-	-	-	-	0.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1
2	2	2,219	0.41	0.04	-	-	-	-	6.0	38.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Ore
2	4	789	0.47	0.06	-	-	-	-	-	18.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	187
2	6	2,505	0.54	0.07	-	-	-	-	-	47.2	18.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Total
2	8	2,162	0.51	0.09	-	-	-	-	-	-	54.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	188
2	10	103	0.65	0.11	-	-	-	-	-	-	3.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	12	23	0.49	0.04	-	-	-	-	-	-	0.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	**GrandTotal**	13,155	0.31	0.04	-	-	-	-	6.9	104.3	76.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	Average,nowaste		0.49	0.06																							
3	1	13,843	0.08	0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
3	3	9	0.27	0.05	-	-	-	-	-	-	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
3	5	36	0.30	0.02	-	-	-	-	-	-	0.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Oxides
3	7	63	0.41	0.05	-	-	-	-	-	-	1.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4
3	9	63	0.59	0.14	-	-	-	-	-	-	0.9	0.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Ore
3	2	2,379	0.38	0.03	-	-	-	-	-	-	-	-	43.9	-	-	-	-	-	-	-	-	-	-	-	-	-	560
3	4	6,908	0.49	0.08	-	-	-	-	-	-	32.6	107.3	26.0	-	-	-	-	-	-	-	-	-	-	-	-	-	Total
3	6	6,085	0.36	0.05	-	-	-	-	-	-	-	-	18.3	79.2	9.6	-	-	-	-	-	-	-	-	-	-	-	564
3	8	11,907	0.41	0.08	-	-	-	-	-	-	-	-	-	-	78.3	89.2	68.4	-	-	-	-	-	-	-	-	-	
3	10	286	0.55	0.04	-	-	-	-	-	-	-	-	-	-	-	-	7.7	-	-	-	-	-	-	-	-	-	
3	12	1,624	0.45	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	35.8	-	-	-	-	-	-	-	-	
3	**GrandTotal**	43,204	0.31	0.05	-	-	-	-	-	-	35.4	108.2	88.2	79.2	87.9	89.2	76.1	35.8	-	-	-	-	-	-	-	-	
	Average,nowaste		0.42	0.07																							
4	1	13,883	0.03	0.00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
4	7	45	0.27	0.01	-	-	-	-	-	-	-	-	-	-	-	-	0.6	-	-	-	-	-	-	-	-	-	-
4	2	2,253	0.31	0.03	-	-	-	-	-	-	-	-	-	-	-	-	11.0	23.6	-	-	-	-	-	-	-	-	Oxides
4	4	2,105	0.46	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	30.1	17.3	-	-	-	-	-	-	-	1
4	6	5,845	0.33	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	59.6	33.7	-	-	-	-	-	-	Ore
4	8	8,704	0.42	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	48.9	92.1	37.2	-	-	-	-	384
4	10	103	0.66	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3.3	-	-	-	-	Total
4	12	824	0.68	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	27.5	-	-	-	-	385
4	**GrandTotal**	33,761	0.25	0.03	-	-	-	-	-	-	-	-	-	-	-	-	11.6	53.7	76.9	82.6	92.1	67.9	-	-	-	-	
	Average,nowaste		0.40	0.05																							
5	1	20,181	0.01	0.00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
5	3	9	0.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.1	-	-	-	-	-
5	2	1,624	0.31	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	24.7	-	-	-	-	Oxides
5	4	1,418	0.44	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2.7	27.5	-	-	-	0
5	6	7,469	0.38	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	59.2	78.7	-	-	Ore
5	8	7,629	0.42	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4.9	92.2	59.2	372
5	10	320	0.52	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	8.1	Total
5	12	515	0.59	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	14.9	372
5	**GrandTotal**	39,165	0.20	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	27.4	86.7	83.5	92.2	82.2	
	Average,nowaste		0.40	0.04																							
grandtotal		159,397	0.4533	0.06																							
			Ore,	**M$**	-	117.0	133.5	135.7	141.5	104.3	109.6	107.3	88.2	79.2	87.9	89.2	87.1	89.5	76.9	82.6	92.1	95.3	86.7	83.5	92.2	82.2	2,061.4
			Oxides,	**M$**	1.8	4.7	-	-	0.9	-	2.8	0.9	-	-	-	-	0.6	-	-	-	-	0.1	-	-	-	-	11.8
			Total,	**M$**	1.8	121.7	133.5	135.7	142.4	104.3	112.3	108.2	88.2	79.2	87.9	89.2	87.6	89.5	76.9	82.6	92.1	95.4	86.7	83.5	92.2	82.2	2,073.2
			Accumulative,	**M$**	1.8	123.5	257.0	392.7	535.1	639.4	751.8	860.0	948.2	1,027.4	1,115.3	1,204.5	1,292.1	1,381.6	1,458.5	1,541.1	1,633.1	1,728.5	1,815.2	1,898.7	1,990.9	2,073.2	-

Table A17 - 7 Processing costs, M$, case scenario 1

Pit stage	*Material Name*	*Tonnes, kt*	*Cu, %*	*Au, g/t*	*I.1*	*II.1*	*III.1*	*IV.1*	*I.2*	*II.2*	*III.2*	*IV.2*	*I.3*	*II.3*	*III.3*	*IV.3*	*I.4*	*II.4*	*III.4*	*IV.4*	*I.5*	*II.5*	*III.5*	*IV.5*	*I.6*	*II.6*	*Control*
1	1	12,454	0.11	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
1	3	72	0.53	0.12	0.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
1	5	18	0.74	0.20	-	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Oxides
1	7	108	0.66	0.12	-	1.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3
1	9	9	0.67	-	-	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Ore
1	11	9	0.63	-	-	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	165
1	2	3,694	0.56	0.07	-	31.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Total
1	4	572	0.59	0.10	-	5.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	167
1	6	8,464	0.61	0.08	-	-	42.7	37.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	8	3,225	0.69	0.07	-	-	-	5.4	27.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	10	1,487	0.61	0.09	-	-	-	-	14.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	**GrandTotal**	30,112	0.40	0.05	0.8	39.0	42.7	43.0	42.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	Average,nowaste		0.61	0.08																							
2	1	5,327	0.03	0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
2	3	9	0.75	0.09	-	-	-	-	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
2	5	9	0.59	0.01	-	-	-	-	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Oxides
2	9	9	0.66	0.13	-	-	-	-	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0
2	2	2,219	0.41	0.04	-	-	-	-	2.6	16.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Ore
2	4	789	0.47	0.06	-	-	-	-	-	7.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	73
2	6	2,505	0.54	0.07	-	-	-	-	-	17.0	6.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Total
2	8	2,162	0.51	0.09	-	-	-	-	-	-	21.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	74
2	10	103	0.65	0.11	-	-	-	-	-	-	1.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	12	23	0.49	0.04	-	-	-	-	-	-	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	**GrandTotal**	13,155	0.31	0.04	-	-	-	-	2.9	40.8	30.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	Average,nowaste		0.49	0.06																							
3	1	13,843	0.08	0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
3	3	9	0.27	0.05	-	-	-	-	-	-	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
3	5	36	0.30	0.02	-	-	-	-	-	-	0.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Oxides
3	7	63	0.41	0.05	-	-	-	-	-	-	0.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2
3	9	63	0.59	0.14	-	-	-	-	-	-	0.4	0.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Ore
3	2	2,379	0.38	0.03	-	-	-	-	-	-	-	-	20.5	-	-	-	-	-	-	-	-	-	-	-	-	-	266
3	4	6,908	0.49	0.08	-	-	-	-	-	-	12.6	41.4	10.0	-	-	-	-	-	-	-	-	-	-	-	-	-	Total
3	6	6,085	0.36	0.05	-	-	-	-	-	-	-	-	9.8	42.7	5.2	-	-	-	-	-	-	-	-	-	-	-	268
3	8	11,907	0.41	0.08	-	-	-	-	-	-	-	-	-	-	40.0	45.6	35.0	-	-	-	-	-	-	-	-	-	
3	10	286	0.55	0.04	-	-	-	-	-	-	-	-	-	-	-	-	2.8	-	-	-	-	-	-	-	-	-	
3	12	1,624	0.45	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	17.2	-	-	-	-	-	-	-	-	
3	**GrandTotal**	43,204	0.31	0.05	-	-	-	-	-	-	14.3	41.8	40.4	42.7	45.2	45.6	37.8	17.2	-	-	-	-	-	-	-	-	
	Average,nowaste		0.42	0.07																							
4	1	13,883	0.03	0.00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
4	7	45	0.27	0.01	-	-	-	-	-	-	-	-	-	-	-	-	0.6	-	-	-	-	-	-	-	-	-	-
4	2	2,253	0.31	0.03	-	-	-	-	-	-	-	-	-	-	-	-	6.2	13.3	-	-	-	-	-	-	-	-	Oxides
4	4	2,105	0.46	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	12.4	7.1	-	-	-	-	-	-	-	1
4	6	5,845	0.33	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	35.4	20.0	-	-	-	-	-	-	Ore
4	8	8,704	0.42	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	24.2	45.6	18.4	-	-	-	-	192
4	10	103	0.66	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.0	-	-	-	-	Total
4	12	824	0.68	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	8.7	-	-	-	-	193
4	**GrandTotal**	33,761	0.25	0.03	-	-	-	-	-	-	-	-	-	-	-	-	6.7	25.7	42.5	44.2	45.6	28.1	-	-	-	-	
	Average,nowaste		0.40	0.05																							
5	1	20,181	0.01	0.00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
5	3	9	0.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.1	-	-	-	-	-
5	2	1,624	0.31	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	14.0	-	-	-	-	Oxides
5	4	1,418	0.44	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.2	12.0	-	-	-	0
5	6	7,469	0.38	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	30.4	40.4	-	-	Ore
5	8	7,629	0.42	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2.4	45.6	29.3	184
5	10	320	0.52	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3.2	Total
5	12	515	0.59	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.4	184
5	**GrandTotal**	39,165	0.20	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	15.3	42.4	42.8	45.6	37.9	
	Average,nowaste		0.40	0.04																							
	GrandGrandtotal		-	0.06																							
grandtotal		159,397	0.4533	0.06																							-
			Ore,	**M$**	-	37.1	42.7	43.0	44.5	40.8	42.5	41.4	40.4	42.7	45.2	45.6	44.0	42.8	42.5	44.2	45.6	43.3	42.4	42.8	45.6	37.9	897.1
			Oxides,	**M$**	0.8	1.8	-	-	0.3	-	1.8	0.4	-	-	-	-	0.6	-	-	-	-	0.1	-	-	-	-	5.9
			Total,	**M$**	0.8	39.0	42.7	43.0	44.9	40.8	44.3	41.8	40.4	42.7	45.2	45.6	44.6	42.8	42.5	44.2	45.6	43.4	42.4	42.8	45.6	37.9	903.0
			Accumulative,	**M$**	0.8	39.8	82.5	125.5	170.4	211.2	255.5	297.4	337.8	380.4	425.7	471.2	515.8	558.6	601.1	645.3	690.9	734.3	776.7	819.5	865.1	903.0	-

Table A17 - 8 Mining costs, M$, case scenario 1

Pit stage	*Material Name*	*Tonnes, kt*	*Cu, %*	*Au, g/t*	*I.1*	*II.1*	*III.1*	*IV.1*	*I.2*	*II.2*	*III.2*	*IV.2*	*I.3*	*II.3*	*III.3*	*IV.3*	*I.4*	*II.4*	*III.4*	*IV.4*	*I.5*	*II.5*	*III.5*	*IV.5*	*I.6*	*II.6*	*Control*
1	1	12,454	0.11	0.02	9.8	3.2	3.3	3.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
1	3	72	0.53	0.12	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	20
1	5	18	0.74	0.20	-	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Oxides
1	7	108	0.66	0.12	-	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0
1	9	9	0.67	-	-	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Ore
1	11	9	0.63	-	-	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	27
1	2	3,694	0.56	0.07	-	5.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Total
1	4	572	0.59	0.10	-	0.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	47
1	6	8,464	0.61	0.08	-	-	7.1	6.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	8	3,225	0.69	0.07	-	-	-	0.8	4.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	10	1,487	0.61	0.09	-	-	-	-	2.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	**GrandTotal**	30,112	0.40	0.05	9.9	10.2	10.4	10.3	6.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	Average,nowaste		0.61																								
2	1	5,327	0.03	0.01	-	-	-	0.2	4.1	4.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Waste
2	3	9	0.75	0.09	-	-	-	-	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	8
2	5	9	0.59	0.01	-	-	-	-	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Oxides
2	9	9	0.66	0.13	-	-	-	-	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0
2	2	2,219	0.41	0.04	-	-	-	-	0.5	3.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Ore
2	4	789	0.47	0.06	-	-	-	-	-	1.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	12
2	6	2,505	0.54	0.07	-	-	-	-	-	2.8	1.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Total
2	8	2,162	0.51	0.09	-	-	-	-	-	-	3.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	21
2	10	103	0.65	0.11	-	-	-	-	-	-	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	12	23	0.49	0.04	-	-	-	-	-	-	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	**GrandTotal**	13,155	0.31	0.04	-	-	-	0.2	4.6	11.2	4.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	Average,nowaste		0.49	0.06																							
3	1	13,843	0.08	0.01	-	-	-	-	-	-	4.3	4.3	4.3	4.3	4.3	0.1	-	-	-	-	-	-	-	-	-	-	Waste
3	3	9	0.27	0.05	-	-	-	-	-	-	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	22
3	5	36	0.30	0.02	-	-	-	-	-	-	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Oxides
3	7	63	0.41	0.05	-	-	-	-	-	-	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0
3	9	63	0.59	0.14	-	-	-	-	-	-	0.0	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	Ore
3	2	2,379	0.38	0.03	-	-	-	-	-	-	-	-	3.7	-	-	-	-	-	-	-	-	-	-	-	-	-	43
3	4	6,908	0.49	0.08	-	-	-	-	-	-	2.1	7.0	1.7	-	-	-	-	-	-	-	-	-	-	-	-	-	Total
3	6	6,085	0.36	0.05	-	-	-	-	-	-	-	-	1.6	7.1	0.9	-	-	-	-	-	-	-	-	-	-	-	65
3	8	11,907	0.41	0.08	-	-	-	-	-	-	-	-	-	-	6.2	7.1	5.4	-	-	-	-	-	-	-	-	-	
3	10	286	0.55	0.04	-	-	-	-	-	-	-	-	-	-	-	-	0.4	-	-	-	-	-	-	-	-	-	
3	12	1,624	0.45	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	2.5	-	-	-	-	-	-	-	-	
3	**GrandTotal**	43,204	0.31	0.05	-	-	-	-	-	-	6.7	11.4	11.4	11.4	11.4	7.2	5.9	2.5	-	-	-	-	-	-	-	-	
	Average,nowaste		0.42	0.07																							
4	1	13,883	0.03	0.00	-	-	-	-	-	-	-	-	-	-	-	4.2	4.4	4.4	4.4	4.4	0.0	-	-	-	-	-	Waste
4	7	45	0.27	0.01	-	-	-	-	-	-	-	-	-	-	-	-	0.1	-	-	-	-	-	-	-	-	-	22
4	2	2,253	0.31	0.03	-	-	-	-	-	-	-	-	-	-	-	-	1.1	2.4	-	-	-	-	-	-	-	-	Oxides
4	4	2,105	0.46	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	2.1	1.2	-	-	-	-	-	-	-	0
4	6	5,845	0.33	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.9	3.3	-	-	-	-	-	-	Ore
4	8	8,704	0.42	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3.7	7.1	2.9	-	-	-	-	31
4	10	103	0.66	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.2	-	-	-	-	Total
4	12	824	0.68	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.3	-	-	-	-	53
4	**GrandTotal**	33,761	0.25	0.03	-	-	-	-	-	-	-	-	-	-	-	4.2	5.6	8.9	11.5	11.5	7.1	4.3	-	-	-	-	
	Average,nowaste		0.40	0.05																							
5	1	20,181	0.01	0.00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.0	5.3	5.7	6.0	6.3	3.5	Waste
5	3	9	0.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.0	-	-	-	-	32
5	2	1,624	0.31	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2.5	-	-	-	-	Oxides
5	4	1,418	0.44	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.2	2.0	-	-	-	0
5	6	7,469	0.38	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.0	6.7	-	-	Ore
5	8	7,629	0.42	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.4	7.1	4.5	30
5	10	320	0.52	0.07	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.5	Total
5	12	515	0.59	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.8	61
5	**GrandTotal**	39,165	0.20	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.0	8.1	12.7	13.0	13.3	9.3	
	Average,nowaste		0.40	0.04																							
	GrandGrandtotal		-	0.06																							
grandtotal		159,397	0.4533	0.06																							
			Ore,	**M$**	-	6.7	7.1	7.1	7.0	7.1	6.8	7.0	7.1	7.1	7.1	7.1	7.0	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	5.9	146.4
			Oxides,	**M$**	0.1	0.2	-	-	0.0	-	0.2	0.1	-	-	-	-	0.1	-	-	-	-	0.0	-	-	-	-	0.7
			Total,	**M$**	0.1	6.9	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	5.9	147.1
			Accumulative,	**M$**	0.1	7.0	14.1	21.2	28.2	35.3	42.4	49.4	56.5	63.6	70.6	77.7	84.8	91.8	98.9	105.9	113.0	120.1	127.1	134.2	141.3	147.1	-

Table A17 - 9 Financial analysis, case scenario 1

		0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
		IV.0	**I.1**	**II.1**	**III.1**	**IV.1**	**I.2**	**II.2**	**III.2**	**IV.2**	**I.3**	**II.3**	**III.3**	**IV.3**	**I.4**	**II.4**	**III.4**	**IV.4**	**I.5**	**II.5**	**III.5**	**IV.5**	**I.6**	**II.6**
Material																								
waste,kt		-	6,250	2,070	2,100	2,134	2,614	2,614	2,750	2,750	2,750	2,750	2,750	2,750	2,800	2,800	2,800	2,800	3,200	3,400	3,600	3,800	4,000	2,207
ore,kt		-	-	4,266	4,500	4,500	4,473	4,500	4,362	4,467	4,500	4,500	4,500	4,500	4,455	4,500	4,500	4,500	4,500	4,491	4,500	4,500	4,500	3,727
oxides,kt		-	72	144	-	-	27	-	138	33	-	-	-	-	45	-	-	-	-	9	-	-	-	-
ore+oxides,kt		-	72	4,410	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	3,727
Total,kt		-	6,322	6,480	6,600	6,634	7,114	7,114	7,250	7,250	7,250	7,250	7,250	7,250	7,300	7,300	7,300	7,300	7,700	7,900	8,100	8,300	8,500	5,934
Me																								
Cu,%		-	0.53	0.57	0.61	0.62	0.65	0.47	0.51	0.49	0.40	0.36	0.40	0.41	0.40	0.41	0.35	0.38	0.42	0.43	0.39	0.38	0.42	0.45
Cu,kt		-	*0.32*	*21.19*	*23.24*	*23.64*	*24.80*	*18.16*	*19.57*	*18.85*	*15.35*	*13.79*	*15.32*	*15.53*	*15.26*	*15.59*	*13.39*	*14.38*	*16.04*	*16.61*	*15.10*	*14.55*	*16.06*	*14.32*
Au,%		-	0.12	0.08	0.08	0.08	0.07	0.05	0.08	0.08	0.05	0.05	0.08	0.08	0.07	0.04	0.03	0.05	0.06	0.05	0.04	0.03	0.05	0.05
Au,kg		-	*5.56*	*217.29*	*243.11*	*238.04*	*217.32*	*159.47*	*240.70*	*242.11*	*140.27*	*140.18*	*219.95*	*230.99*	*199.70*	*114.48*	*98.99*	*132.49*	*171.22*	*142.19*	*121.77*	*99.71*	*141.15*	*119.73*
Revenues																								
Revenue,k$	**2.8**	**0**	1,987.8	130,809.3	143,488.9	145,917.4	153,097.0	112,118.1	120,776.1	116,368.5	94,782.4	85,123.9	94,539.0	95,842.7	94,216.6	96,213.6	82,644.7	88,791.8	98,983.5	102,508.9	93,219.2	89,790.8	99,119.3	88,411.2
Depriciation	**771,429**	**0**	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184
EBIT,kt		**-**	**-7,196**	**121,626**	**134,305**	**136,734**	**143,913**	**102,934**	**111,592**	**107,185**	**85,599**	**75,940**	**85,355**	**86,659**	**85,033**	**87,030**	**73,461**	**79,608**	**89,800**	**93,325**	**84,036**	**80,607**	**89,936**	**79,228**
Costs																								
MainCosts																								
milling,k$		-	836	38,981	42,674	43,020	44,879	40,846	44,293	41,842	40,389	42,674	45,229	45,583	44,557	42,810	42,508	44,218	45,583	43,372	42,395	42,827	45,583	37,903
mining,k$		-	113	6,924	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	5,851
Capex		**771,429**																						
Total,k$		**771,429**	**949**	**45,905**	**49,739**	**50,085**	**51,944**	**47,911**	**51,358**	**48,907**	**47,454**	**49,739**	**52,294**	**52,648**	**51,622**	**49,875**	**49,573**	**51,283**	**52,648**	**50,437**	**49,460**	**49,892**	**52,648**	**43,754**
	$/tOpex		**5.46**	**1.26**	**1.24**	**1.23**	**1.22**	**1.49**	**1.48**	**1.47**	**1.71**	**1.95**	**1.85**	**1.84**	**1.84**	**1.76**	**2.00**	**1.93**	**1.80**	**1.68**	**1.80**	**1.88**	**1.80**	**1.68**
AdditionalCosts																								
Extractiontax	**8%**	**-**	**159**	**10,465**	**11,479**	**11,673**	**12,248**	**8,969**	**9,662**	**9,309**	**7,583**	**6,810**	**7,563**	**7,667**	**7,537**	**7,697**	**6,612**	**7,103**	**7,919**	**8,201**	**7,458**	**7,183**	**7,930**	**7,073**
Amortization$/t	**0.09**	**-**	**579**	**594**	**605**	**608**	**652**	**652**	**664**	**664**	**664**	**664**	**664**	**664**	**669**	**669**	**669**	**669**	**706**	**724**	**742**	**761**	**779**	**544**
Generalproductioncosts,$/m3	**0.35**	**-**	**2,221**	**2,276**	**2,318**	**2,330**	**2,499**	**2,499**	**2,547**	**2,547**	**2,547**	**2,547**	**2,547**	**2,547**	**2,564**	**2,564**	**2,564**	**2,564**	**2,705**	**2,775**	**2,845**	**2,915**	**2,986**	**2,084**
TAX,%	**0%**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Transport,$/tconc	**-**		**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Total,k$		**-**	**2,959**	**13,335**	**14,402**	**14,611**	**15,398**	**12,120**	**12,873**	**12,520**	**10,793**	**10,021**	**10,774**	**10,878**	**10,770**	**10,930**	**9,845**	**10,336**	**11,329**	**11,699**	**11,045**	**10,859**	**11,694**	**9,701**
Grandtotal,K$		**771,429**	**3,908**	**59,239**	**64,141**	**64,697**	**67,343**	**60,031**	**64,231**	**61,428**	**58,248**	**59,760**	**63,068**	**63,527**	**62,393**	**60,805**	**59,418**	**61,619**	**63,977**	**62,136**	**60,505**	**60,751**	**64,342**	**53,455**
Profits																								
CF,k$		-771,429	-1,920	71,570	79,348	81,221	85,755	52,087	56,545	54,941	36,535	25,364	31,471	32,316	31,824	35,409	23,227	27,172	35,006	40,373	32,714	29,040	34,777	34,956
Accum,k$		-771,429	-773,349	-701,779	-622,431	-541,210	-455,455	-403,369	-346,823	-291,882	-255,348	-229,983	-198,513	-166,197	-134,373	-98,964	-75,737	-48,565	-13,558	26,814	59,529	88,568	123,345	158,302
Discounted																								
Discountrate	**2.50%**																							
DCF,k$		-771,429	-1,873	68,121	73,683	73,582	75,795	44,914	47,570	45,093	29,254	19,815	23,985	24,029	23,086	25,060	16,037	18,304	23,006	25,886	20,464	17,722	20,706	20,305
NPV,k$		-771,429	-773,302	-705,181	-631,498	-557,916	-482,121	-437,207	-389,638	-344,545	-315,291	-295,476	-271,491	-247,462	-224,376	-199,316	-183,279	-164,975	-141,969	-116,084	-95,620	-77,898	-57,192	-36,887
	5years	**total**			**5years**	**total**			**5years**	**total**			**5years**	**total**										
IRR	1.95%	4.88%																						
paybackDCF,q	20.3	30.9	**paybackDCF,y**		5.1	7.7	**paybackCF,q**		16.7	19.5	**paybackCF,y**		4.2	4.9										

Table A18 - 1 Mine schedule, case scenario 2

Pit stage	*Material Name*	*Tonnes, kt*	*Cu, %*	*Au, g/t*	*I.1*	*II.1*	*III.1*	*IV.1*	*I.2*	*II.2*	*III.2*	*IV.2*	*I.3*	*II.3*	*III.3*	*IV.3*	*I.4*	*II.4*	*III.4*	*IV.4*	*I.5*	*II.5*	*III.5*	*IV.5*	*I.6*	*II.6*	*Control*
1	A_oxidation	12,454	0.11	0.02	6,250	2,070	2,100	2,034																			-
1	B_oxidation	216	0.62	0.11	72	144	-	-																			-
1	Fresh	17,443	0.61	0.08	-	4,266	4,500	4,500	4,176																		-
1	Total	30,112	0.40	0.05																							
2	A_oxidation	5,327	0.03	0.01				100	2,614	2,614																	-
2	B_oxidation	27	0.67	0.08					27	-	-																-
2	Fresh	7,801	0.49	0.06					297	4,500	3,004																-
2	Total	13,155	0.31	0.04																							
3	A_oxidation	13,843	0.08	0.01							2,750	2,750	2,750	2,750	2,750	93											-
3	B_oxidation	171	0.45	0.08							138	33	-	-	-	-											-
3	Fresh	29,189	0.42	0.07							1,358	4,467	4,500	4,500	4,500	4,500	3,740	1,624									-
3	Total	43,204	0.31	0.05																							
4	A_oxidation	13,883	0.03	0.00												2,657	2,800	2,800	2,800	2,800	26						-
4	B_oxidation	45	0.27	0.01												-	45	-	-	-	-	-					-
4	Fresh	19,833	0.40	0.05												-	715	2,876	4,500	4,500	4,500	2,742					-
4	Total	33,761	0.25	0.03																							
5	A_oxidation	20,181	0.01	0.00																	3,174	3,400	3,600	3,800	4,000	2,207	-
5	B_oxidation	9	0.16	-																	-	9	-	-	-	-	-
5	Fresh	18,975	0.40	0.04																	-	1,749	4,500	4,500	4,500	3,727	-
5	Total	39,165	0.20	0.02																							
waste, kt					6,250	2,070	2,100	2,134	2,614	2,614	2,750	2,750	2,750	2,750	2,750	2,750	2,800	2,800	2,800	2,800	3,200	3,400	3,600	3,800	4,000	2,207	**65,689**
ore, kt					-	4,266	4,500	4,500	4,473	4,500	4,362	4,467	4,500	4,500	4,500	4,500	4,455	4,500	4,500	4,500	4,500	4,491	4,500	4,500	4,500	3,727	**93,241**
transitional, kt					72	144	-	-	27	-	138	33	-	-	-	-	45	-	-	-	-	9	-	-	-	-	**468**
ore+oxides, kt					72	4,410	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	3,727	**93,709**
total, kt					6,322	6,480	6,600	6,634	7,114	7,114	7,250	7,250	7,250	7,250	7,250	7,250	7,300	7,300	7,300	7,300	7,700	7,900	8,100	8,300	8,500	5,934	**159,397**

Table A18 - 2 Cu grades, case scenario 2

Pit stage	*Material Name*	*Tonnes, kt*	*Cu, %*	*Au, g/t*	*I.1*	*II.1*	*III.1*	*IV.1*	*I.2*	*II.2*	*III.2*	*IV.2*	*I.3*	*II.3*	*III.3*	*IV.3*	*I.4*	*II.4*	*III.4*	*IV.4*	*I.5*	*II.5*	*III.5*	*IV.5*	*I.6*	*II.6*	*Control*
1	A_oxidation	12,454	0.11	0.02	0.11	0.11	0.11	0.11	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.11
1	B_oxidation	216	0.62	0.11	0.62	0.62	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.62
1	Fresh	17,443	0.61	0.08	-	0.61	0.61	0.61	0.61	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.61
1	Total	30,112	0.40	0.05																							
2	A_oxidation	5,327	0.03	0.01	-	-	-	0.03	0.03	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.03
2	B_oxidation	27	0.67	0.08	-	-	-	-	0.67	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.67
2	Fresh	7,801	0.49	0.06	-	-	-	-	0.49	0.49	0.49	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.49
2	Total	13,155	0.31	0.04																							
3	A_oxidation	13,843	0.08	0.01	-	-	-	-	-	-	0.08	0.08	0.08	0.08	0.08	0.08	-	-	-	-	-	-	-	-	-	-	0.08
3	B_oxidation	171	0.45	0.08	-	-	-	-	-	-	0.45	0.45	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.45
3	Fresh	29,189	0.42	0.07	-	-	-	-	-	-	0.42	0.42	0.42	0.42	0.42	0.42	0.42	0.42	-	-	-	-	-	-	-	-	0.42
3	Total	43,204	0.31	0.05																							
4	A_oxidation	13,883	0.03	0.00	-	-	-	-	-	-	-	-	-	-	-	0.03	0.03	0.03	0.03	0.03	0.03	-	-	-	-	-	0.03
4	B_oxidation	45	0.27	0.01	-	-	-	-	-	-	-	-	-	-	-	-	0.27	-	-	-	-	-	-	-	-	-	0.27
4	Fresh	19,833	0.40	0.05	-	-	-	-	-	-	-	-	-	-	-	-	0.40	0.40	0.40	0.40	0.40	0.40	-	-	-	-	0.40
4	Total	33,761	0.25	0.03																							
5	A_oxidation	20,181	0.01	0.00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.01	0.01	0.01	0.01	0.01	0.01	0.01
5	B_oxidation	9	0.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.16	-	-	-	-	0.16
5	Fresh	18,975	0.40	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.40	0.40	0.40	0.40	0.40	0.40
5	Total	39,165	0.20	0.02																							
	(Grand total)	(159,397)	(0.45)	(0.06)																							
	average, %				0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	
	max, %				0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	
	real, %				0.62	0.61	0.61	0.61	0.61	0.49	0.47	0.42	0.42	0.42	0.42	0.42	0.41	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.453
	min, %				0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	

Table A18 - 3 Cu tonnage, case scenario 2

Pit stage	*Material Name*	*Tonnes, kt*	*Cu, %*	*Au, g/t*	*I.1*	*II.1*	*III.1*	*IV.1*	*I.2*	*II.2*	*III.2*	*IV.2*	*I.3*	*II.3*	*III.3*	*IV.3*	*I.4*	*II.4*	*III.4*	*IV.4*	*I.5*	*II.5*	*III.5*	*IV.5*	*I.6*	*II.6*	*Control*
1	A_oxidation	12,454	0.11	0.02																							-
1	B_oxidation	216	0.62	0.11	0.38	0.76	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.14
1	Fresh	17,443	0.61	0.08	-	22.22	23.44	23.44	21.76	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	90.86
1	Total	30,112	0.40	0.05																							
2	A_oxidation	5,327	0.03	0.01																							-
2	B_oxidation	27	0.67	0.08	-	-	-	-	0.15	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.15
2	Fresh	7,801	0.49	0.06	-	-	-	-	1.24	18.81	12.56	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	32.61
2	Total	13,155	0.31	0.04																							
3	A_oxidation	13,843	0.08	0.01																							-
3	B_oxidation	171	0.45	0.08	-	-	-	-	-	-	0.52	0.13	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.65
3	Fresh	29,189	0.42	0.07	-	-	-	-	-	-	4.83	15.89	16.01	16.01	16.01	16.01	13.30	5.78	-	-	-	-	-	-	-	-	103.84
3	Total	43,204	0.31	0.05																							
4	A_oxidation	13,883	0.03	0.00																							-
4	B_oxidation	45	0.27	0.01	-	-	-	-	-	-	-	-	-	-	-	-	0.10	-	-	-	-	-	-	-	-	-	0.10
4	Fresh	19,833	0.40	0.05	-	-	-	-	-	-	-	-	-	-	-	-	2.41	9.70	15.18	15.18	15.18	9.25	-	-	-	-	66.90
4	Total	33,761	0.25	0.03																							
5	A_oxidation	20,181	0.01	0.00																							-
5	B_oxidation	9	0.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.01	-	-	-	-	0.01
5	Fresh	18,975	0.40	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.97	15.37	15.37	15.37	12.73	64.79
5	Total	39,165	0.20	0.02																							
	(Grand total)	(159,397)	(0.45)	(0.06)																							
	Cu, kt				0.38	22.98	23.44	23.44	23.15	18.81	17.91	16.02	16.01	16.01	16.01	16.01	15.82	15.48	15.18	15.18	15.18	15.23	15.37	15.37	15.37	12.73	**361.05**
	Con, kt				1.74	105.43	107.53	107.53	106.18	86.28	82.17	73.47	73.43	73.43	73.43	73.43	72.56	71.00	69.63	69.63	69.63	69.88	70.48	70.48	70.48	58.37	**1,656.20**

Table A18 - 4 Au grades, case scenario 2

Pit stage	*Material Name*	*Tonnes, kt*	*Cu, %*	*Au, g/t*	*I.1*	*II.1*	*III.1*	*IV.1*	*I.2*	*II.2*	*III.2*	*IV.2*	*I.3*	*II.3*	*III.3*	*IV.3*	*I.4*	*II.4*	*III.4*	*IV.4*	*I.5*	*II.5*	*III.5*	*IV.5*	*I.6*	*II.6*	*Control*
1	A_oxidation	12,454	0.11	0.02	0.02	0.02	0.02	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.02
1	B_oxidation	216	0.62	0.11	0.11	0.11	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.11
1	Fresh	17,443	0.61	0.08	-	0.08	0.08	0.08	0.08	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.08
1	Total	30,112	0.40	0.05																							
2	A_oxidation	5,327	0.03	0.01	-	-	-	0.01	0.01	0.01	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.01
2	B_oxidation	27	0.67	0.08	-	-	-	-	0.08	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.08
2	Fresh	7,801	0.49	0.06	-	-	-	-	0.06	0.06	0.06	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.06
2	Total	13,155	0.31	0.04																							
3	A_oxidation	13,843	0.08	0.01	-	-	-	-	-	-	0.01	0.01	0.01	0.01	0.01	0.01	-	-	-	-	-	-	-	-	-	-	0.01
3	B_oxidation	171	0.45	0.08	-	-	-	-	-	-	0.08	0.08	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.08
3	Fresh	29,189	0.42	0.07	-	-	-	-	-	-	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	-	-	-	-	-	-	-	-	0.07
3	Total	43,204	0.31	0.05																							
4	A_oxidation	13,883	0.03	0.00	-	-	-	-	-	-	-	-	-	-	-	0.00	0.00	0.00	0.00	0.00	0.00	-	-	-	-	-	0.00
4	B_oxidation	45	0.27	0.01	-	-	-	-	-	-	-	-	-	-	-	-	0.01	-	-	-	-	-	-	-	-	-	0.01
4	Fresh	19,833	0.40	0.05	-	-	-	-	-	-	-	-	-	-	-	-	0.05	0.05	0.05	0.05	0.05	0.05	-	-	-	-	0.05
4	Total	33,761	0.25	0.03																							
5	A_oxidation	20,181	0.01	0.00	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.00	0.00	0.00	0.00	0.00	0.00	0.00
5	B_oxidation	9	0.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
5	Fresh	18,975	0.40	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.04	0.04	0.04	0.04	0.04	0.04
5	Total	39,165	0.20	0.02																							
	(Grand total)	(159,397)	(0.45)	(0.06)																							
	Au, g/t				0.11	0.08	0.08	0.08	0.08	0.06	0.07	0.07	0.07	0.07	0.07	0.07	0.06	0.05	0.05	0.05	0.05	0.04	0.04	0.04	0.04	0.04	**0.06**

Table A18 - 5 Au mass, case scenario 2

Pit stage	*Material Name*	*Tonnes, kt*	*Cu, %*	*Au, g/t*	*I.1*	*II.1*	*III.1*	*IV.1*	*I.2*	*II.2*	*III.2*	*IV.2*	*I.3*	*II.3*	*III.3*	*IV.3*	*I.4*	*II.4*	*III.4*	*IV.4*	*I.5*	*II.5*	*III.5*	*IV.5*	*I.6*	*II.6*	*Control*
1	A_oxidation	12,454	0.11	0.02																							-
1	B_oxidation	216	0.62	0.11	5.37	10.75	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	16.12
1	Fresh	17,443	0.61	0.08	-	219.27	231.28	231.28	214.64	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	896.46
1	Total	30,112	0.40	0.05																							
2	A_oxidation	5,327	0.03	0.01																							-
2	B_oxidation	27	0.67	0.08	-	-	-	-	1.35	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.35
2	Fresh	7,801	0.49	0.06	-	-	-	-	12.52	189.89	126.76	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	329.16
2	Total	13,155	0.31	0.04																							
3	A_oxidation	13,843	0.08	0.01																							-
3	B_oxidation	171	0.45	0.08	-	-	-	-	-	-	6.97	1.66	-	-	-	-	-	-	-	-	-	-	-	-	-	-	8.63
3	Fresh	29,189	0.42	0.07	-	-	-	-	-	-	58.89	193.70	195.13	195.13	195.13	195.13	162.18	70.43	-	-	-	-	-	-	-	-	1,265.72
3	Total	43,204	0.31	0.05																							
4	A_oxidation	13,883	0.03	0.00																							-
4	B_oxidation	45	0.27	0.01	-	-	-	-	-	-	-	-	-	-	-	-	0.15	-	-	-	-	-	-	-	-	-	0.15
4	Fresh	19,833	0.40	0.05	-	-	-	-	-	-	-	-	-	-	-	-	21.26	85.49	133.77	133.77	133.77	81.52	-	-	-	-	589.60
4	Total	33,761	0.25	0.03																							
5	A_oxidation	20,181	0.01	0.00																							-
5	B_oxidation	9	0.16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
5	Fresh	18,975	0.40	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	48.77	125.51	125.51	125.51	103.94	529.24
5	Total	39,165	0.20	0.02																							
	(Grand total)	(159,397)	(0.45)	(0.06)																							
	Au, kg				5.37	230.01	231.28	231.28	228.51	189.89	192.62	195.37	195.13	195.13	195.13	195.13	183.58	155.92	133.77	133.77	133.77	130.30	125.51	125.51	125.51	103.94	**3,636.42**

Table A18 - 6 Revenues, case scenario 2

NSR	*48807.2 $/kt*															*Revenues*											
Pit stage	**Material Name**	**Tonnes, kt**	**Cu, %**	**Au, g/t**	I.1	II.1	III.1	IV.1	I.2	II.2	III.2	IV.2	I.3	II.3	III.3	IV.3	I.4	II.4	III.4	IV.4	I.5	II.5	III.5	IV.5	I.6	II.6	Control
					-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	A_oxidation	12,454	0.106	0.019	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	B_oxidation	216	0.62	0.11	2.2	4.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	6.5
1	Ore	17,443	0.61	0.08	-	127.6	134.6	134.6	124.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	521.7
1	Total	30,112	0.40	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	A_oxidation	5,327	0.033	0.007	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	B_oxidation	27	0.67	0.08	-	-	-	-	0.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.9
2	Ore	7,801	0.49	0.06	-	-	-	-	7.1	108.0	72.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	187.2
2	Total	13,155	0.31	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
3	A_oxidation	13,843	0.076	0.008	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
3	B_oxidation	171	0.45	0.08	-	-	-	-	-	-	3.0	0.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3.7
3	Ore	29,189	0.42	0.07	-	-	-	-	-	-	27.7	91.2	91.9	91.9	91.9	91.9	76.4	33.2	-	-	-	-	-	-	-	-	596.2
3	Total	43,204	0.31	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
4	A_oxidation	13,883	0.028	0.004	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
4	B_oxidation	45	0.265	0.005	-	-	-	-	-	-	-	-	-	-	-	-	0.6	-	-	-	-	-	-	-	-	-	0.6
4	Ore	19,833	0.40	0.05	-	-	-	-	-	-	-	-	-	-	-	-	13.8	55.7	87.2	87.2	87.2	53.1	-	-	-	-	384.1
4	Total	33,761	0.25	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
5	A_oxidation	20,181	0.009	0.002	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
5	B_oxidation	9	0.158	0.000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.1	-	-	-	-	0.1
5	Ore	18,975	0.40	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	34.3	88.2	88.2	88.2	73.1	372.0
5	Total	39,165	0.20	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	(Grand total)	(159,397)	(0.45)	(0.06)																							
	Ore, M$					127.6	134.6	134.6	132.0	108.0	99.8	91.2	91.9	91.9	91.9	91.9	90.2	88.9	87.2	87.2	87.2	87.4	88.2	88.2	88.2	73.1	2,061.4
	Oxides, M$					4.4	-	-	0.9	-	3.0	0.7	-	-	-	-	0.6	-	-	-	-	0.1	-	-	-	-	11.8
	Total, M$					132.0	134.6	134.6	132.9	108.0	102.9	92.0	91.9	91.9	91.9	91.9	90.8	88.9	87.2	87.2	87.2	87.5	88.2	88.2	88.2	73.1	2,073.2

Table A18 - 7 Processing costs, case scenario 2

NSR	*48807.2 $/kt*															*Processing cost*											
Pit stage	**Material Name**	**Tonnes, kt**	**Cu, %**	**Au, g/t**	I.1	II.1	III.1	IV.1	I.2	II.2	III.2	IV.2	I.3	II.3	III.3	IV.3	I.4	II.4	III.4	IV.4	I.5	II.5	III.5	IV.5	I.6	II.6	Control
1	A_oxidation	12,454	0.106	0.019	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
1	B_oxidation	216	0.62	0.11	0.9	1.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2.7
1	Ore	17,443	0.61	0.08	-	41.0	43.3	43.3	40.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	167.8
1	Total	30,112	0.40	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	A_oxidation	5,327	0.033	0.007	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	B_oxidation	27	0.67	0.08	-	-	-	-	0.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.3
2	Ore	7,801	0.49	0.06	-	-	-	-	2.9	43.3	28.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	75.1
2	Total	13,155	0.31	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
3	A_oxidation	13,843	0.076	0.008	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
3	B_oxidation	171	0.45	0.08	-	-	-	-	-	-	1.7	0.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2.1
3	Ore	29,189	0.42	0.07	-	-	-	-	-	-	13.1	43.0	43.3	43.3	43.3	43.3	36.0	15.6	-	-	-	-	-	-	-	-	280.8
3	Total	43,204	0.31	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
4	A_oxidation	13,883	0.028	0.004	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
4	B_oxidation	45	0.265	0.005	-	-	-	-	-	-	-	-	-	-	-	-	0.6	-	-	-	-	-	-	-	-	-	0.6
4	Ore	19,833	0.40	0.05	-	-	-	-	-	-	-	-	-	-	-	-	6.9	27.7	43.3	43.3	43.3	26.4	-	-	-	-	190.8
4	Total	33,761	0.25	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
5	A_oxidation	20,181	0.009	0.002	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
5	B_oxidation	9	0.158	0.000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.1	-	-	-	-	0.1
5	Ore	18,975	0.40	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	16.8	43.3	43.3	43.3	35.9	182.6
5	Total	39,165	0.20	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	(Grand total)	(159,397)	(0.45)	(0.06)																							-
	Ore, M$				-	41.0	43.3	43.3	43.0	43.3	42.0	43.0	43.3	43.3	43.3	43.3	42.9	43.3	43.3	43.3	43.3	43.2	43.3	43.3	43.3	35.9	897.1
	Oxides, M$				0.9	1.8	-	-	0.3	-	1.7	0.4	-	-	-	-	0.6	-	-	-	-	0.1	-	-	-	-	5.9
	Total, M$				0.9	42.9	43.3	43.3	43.4	43.3	43.7	43.4	43.3	43.3	43.3	43.3	43.4	43.3	43.3	43.3	43.3	43.3	43.3	43.3	43.3	35.9	903.0
	Accumulative, M$				0.9	43.8	87.1	130.4	173.7	217.0	260.7	304.1	347.4	390.7	434.0	477.3	520.7	564.0	607.3	650.6	693.9	737.3	780.6	823.9	867.1	903.0	-

Table A18 - 8 Mining costs, case scenario 2

NSR	*48807.2 $/kt*															*miningcost*											
Pit stage	**Material Name**	**Tonnes, kt**	**Cu, %**	**Au, g/t**	I.1	II.1	III.1	IV.1	I.2	II.2	III.2	IV.2	I.3	II.3	III.3	IV.3	I.4	II.4	III.4	IV.4	I.5	II.5	III.5	IV.5	I.6	II.6	Control
1	A_oxidation	12,454	0.106	0.019	9.8	3.2	3.3	3.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	19.6
1	B_oxidation	216	0.62	0.11	0.1	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.3
1	Ore	17,443	0.61	0.08	-	6.7	7.1	7.1	6.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	27.4
1	Total	30,112	0.40	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
2	A_oxidation	5,327	0.033	0.007	-	-	-	0.2	4.1	4.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	8.4
2	B_oxidation	27	0.67	0.08	-	-	-	-	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.0
2	Ore	7,801	0.49	0.06	-	-	-	-	0.5	7.1	4.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	12.2
2	Total	13,155	0.31	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
3	A_oxidation	13,843	0.076	0.008	-	-	-	-	-	-	4.3	4.3	4.3	4.3	4.3	0.1	-	-	-	-	-	-	-	-	-	-	21.7
3	B_oxidation	171	0.45	0.08	-	-	-	-	-	-	0.2	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.3
3	Ore	29,189	0.42	0.07	-	-	-	-	-	-	2.1	7.0	7.1	7.1	7.1	7.1	5.9	2.5	-	-	-	-	-	-	-	-	45.8
3	Total	43,204	0.31	0.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
4	A_oxidation	13,883	0.028	0.004	-	-	-	-	-	-	-	-	-	-	-	4.2	4.4	4.4	4.4	4.4	0.0	-	-	-	-	-	21.8
4	B_oxidation	45	0.265	0.005	-	-	-	-	-	-	-	-	-	-	-	-	0.1	-	-	-	-	-	-	-	-	-	0.1
4	Ore	19,833	0.40	0.05	-	-	-	-	-	-	-	-	-	-	-	-	1.1	4.5	7.1	7.1	7.1	4.3	-	-	-	-	31.1
4	Total	33,761	0.25	0.03	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
5	A_oxidation	20,181	0.009	0.002	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.0	5.3	5.7	6.0	6.3	3.5	31.7
5	B_oxidation	9	0.158	0.000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.0	-	-	-	-	0.0
5	Ore	18,975	0.40	0.04	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2.7	7.1	7.1	7.1	5.9	29.8
5	Total	39,165	0.20	0.02	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
	(Grand total)	(159,397)	(0.45)	(0.06)																							-
	Ore, M$				-	6.7	7.1	7.1	7.0	7.1	6.8	7.0	7.1	7.1	7.1	7.1	7.0	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	5.9	146.4
	Oxides, M$				0.1	0.2	-	-	0.0	-	0.2	0.1	-	-	-	-	0.1	-	-	-	-	0.0	-	-	-	-	0.7
	Total, M$				0.1	6.9	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	7.1	5.9	147.1
	Accumulative, M$				0.1	7.0	14.1	21.2	28.2	35.3	42.4	49.4	56.5	63.6	70.6	77.7	84.8	91.8	98.9	105.9	113.0	120.1	127.1	134.2	141.3	147.1	-

Table A18 - 9 Fianancial analysis, case scenario 2

Material		*0*	*1*	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9*	*10*	*11*	*12*	*13*	*14*	*15*	*16*	*17*	*18*	*19*	*20*	*21*	*22*
		IV.0	**I.1**	**II.1**	**III.1**	**IV.1**	**I.2**	**II.2**	**III.2**	**IV.2**	**I.3**	**II.3**	**III.3**	**IV.3**	**I.4**	**II.4**	**III.4**	**IV.4**	**I.5**	**II.5**	**III.5**	**IV.5**	**I.6**	**II.6**
waste,kt		-	6,250	2,070	2,100	2,134	2,614	2,614	2,750	2,750	2,750	2,750	2,750	2,750	2,800	2,800	2,800	2,800	3,200	3,400	3,600	3,800	4,000	2,207
ore,kt		-	-	4,266	4,500	4,500	4,473	4,500	4,362	4,467	4,500	4,500	4,500	4,500	4,455	4,500	4,500	4,500	4,500	4,491	4,500	4,500	4,500	3,727
oxides,kt		-	72	144	-	-	27	-	138	33	-	-	-	-	45	-	-	-	-	9	-	-	-	-
ore+oxides,kt		-	72	4,410	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500	3,727
Total,kt		-	6,322	6,480	6,600	6,634	7,114	7,114	7,250	7,250	7,250	7,250	7,250	7,250	7,300	7,300	7,300	7,300	7,700	7,900	8,100	8,300	8,500	5,934
Me																								
Cu,%		-	0.62	0.61	0.61	0.61	0.61	0.49	0.47	0.42	0.42	0.42	0.42	0.42	0.41	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40
Cu,kt		-	*0.38*	*22.98*	*23.44*	*23.44*	*23.15*	*18.81*	*17.91*	*16.02*	*16.01*	*16.01*	*16.01*	*16.01*	*15.82*	*15.48*	*15.18*	*15.18*	*15.18*	*15.23*	*15.37*	*15.37*	*15.37*	*12.73*
Au,%		-	0.11	0.08	0.08	0.08	0.08	0.06	0.07	0.07	0.07	0.07	0.07	0.07	0.06	0.05	0.05	0.05	0.05	0.04	0.04	0.04	0.04	0.04
Au,kg		-	*5.37*	*230.01*	*231.28*	*231.28*	*228.51*	*189.89*	*192.62*	*195.37*	*195.13*	*195.13*	*195.13*	*195.13*	*183.58*	*155.92*	*133.77*	*133.77*	*133.77*	*130.30*	*125.51*	*125.51*	*125.51*	*103.94*
Revenue,k$		**0**	2,180	131,968	134,598	134,598	132,915	108,007	102,854	91,968	91,921	91,921	91,921	91,921	90,828	88,876	87,156	87,156	87,156	87,468	88,229	88,229	88,229	73,067
Depriciation	**771,429**	**0**	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184	-9,184
EBIT,kt		-	**-7,004**	**122,784**	**125,414**	**125,414**	**123,731**	**98,824**	**93,671**	**82,784**	**82,737**	**82,737**	**82,737**	**82,737**	**81,644**	**79,692**	**77,972**	**77,972**	**77,972**	**78,284**	**79,045**	**79,045**	**79,045**	**63,884**
Costs																								
MainCosts																								
milling,k$		-	904	42,857	43,297	43,297	43,377	43,297	43,703	43,394	43,297	43,297	43,297	43,297	43,429	43,297	43,297	43,297	43,297	43,324	43,297	43,297	43,297	35,857
mining,k$		-	113	6,924	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	7,065	5,851
Capex		**771,429**																						
Total,k$		**771,429**	**1,017**	**49,781**	**50,362**	**50,362**	**50,442**	**50,362**	**50,768**	**50,459**	**50,362**	**50,362**	**50,362**	**50,362**	**50,494**	**50,362**	**50,362**	**50,362**	**50,362**	**50,389**	**50,362**	**50,362**	**50,362**	**41,708**
AdditionalCosts																								
Extractiontax	**8%**	-	**174**	**10,557**	**10,768**	**10,768**	**10,633**	**8,641**	**8,228**	**7,357**	**7,354**	**7,354**	**7,354**	**7,354**	**7,266**	**7,110**	**6,972**	**6,972**	**6,972**	**6,997**	**7,058**	**7,058**	**7,058**	**5,845**
Amortization$/t	**0.09**	-	**579**	**594**	**605**	**608**	**652**	**652**	**664**	**664**	**664**	**664**	**664**	**664**	**669**	**669**	**669**	**669**	**706**	**724**	**742**	**761**	**779**	**544**
Generalproductioncosts,$/m3	**0.35**	-	**2,221**	**2,276**	**2,318**	**2,330**	**2,499**	**2,499**	**2,547**	**2,547**	**2,547**	**2,547**	**2,547**	**2,547**	**2,564**	**2,564**	**2,564**	**2,564**	**2,705**	**2,775**	**2,845**	**2,915**	**2,986**	**2,084**
TAX,%	**0%**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Transport,$/tconc	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total,k$		-	**2,974**	**13,427**	**13,691**	**13,706**	**13,784**	**11,791**	**11,439**	**10,568**	**10,565**	**10,565**	**10,565**	**10,565**	**10,499**	**10,343**	**10,205**	**10,205**	**10,383**	**10,496**	**10,646**	**10,734**	**10,823**	**8,473**
Grandtotal,K$		**771,429**	**3,991**	**63,208**	**64,053**	**64,068**	**64,225**	**62,153**	**62,207**	**61,027**	**60,927**	**60,927**	**60,927**	**60,927**	**60,994**	**60,705**	**60,568**	**60,568**	**60,745**	**60,885**	**61,008**	**61,096**	**61,185**	**50,181**
CF,k$		- 771,429	-1,811	68,760	70,545	70,530	68,690	45,854	40,647	30,940	30,994	30,994	30,994	30,994	29,834	28,171	26,588	26,588	26,411	26,583	27,221	27,133	27,044	22,886
Accum,k$		- 771,429	- 773,240	- 704,480	- 633,935	- 563,405	- 494,715	- 448,861	- 408,214	- 377,273	- 346,279	- 315,285	- 284,290	- 253,296	- 223,462	- 195,291	- 168,703	- 142,114	- 115,703	-89,120	-61,898	-34,766	-7,722	15,164
Discount rate	**2.50%**																							
DCF,k$		- 771,429	-1,767	65,447	65,508	63,897	60,712	39,540	34,195	25,394	24,818	24,213	23,622	23,046	21,642	19,937	18,358	17,911	17,357	17,044	17,028	16,558	16,102	13,294
NPV,k$		- 771,429	- 773,196	- 707,749	- 642,241	- 578,344	- 517,632	- 478,092	- 443,897	- 418,503	- 393,685	- 369,473	- 345,850	- 322,804	- 301,162	- 281,225	- 262,866	- 244,956	- 227,598	- 210,554	- 193,526	- 176,968	- 160,867	- 147,573
	5years	**total**			**5years**	**total**			**5years**	**total**														
IRR	0.20%	4.18%	**paybackDCF,y**		6.0	9.2	**paybackCF,y**		4.9	5.7														
paybackDCF,q	24.1	36.7	**paybackCF,q**		19.8	22.8																		